# STABILITY AND SUPPORT IN DEEP MINES

## PROGRESS REPORT

BY

FALCONBRIDGE LIMITED

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Engineering work upon this project was carried out by the following team:

| | |
|---|---|
| K.H. Singh, Ph.D.,P. Eng. | Project Manager prior to Dec. 1, 1981 |
| M. Musson, B.Sc., P. Eng. | Project Manager from Dec. 1, 1981 |
| S. Vongpaisal, Ph.D., P. Eng. | Chief Geomechanics Engineer |
| D. West, M.Sc., C. Eng. | Geomechanics Engineer - Projects |
| R. W. Jackson, M.Sc., P. Eng. | Geomechanics Engineer - Explosives Technology |
| K. Jansons, B.Sc. | Geomechanics Supervisor |

Additional operating and technical input was provided by the following Falconbridge Limited personnel: R.L. Montgomery, G.V. Hill, T.A. Trott, T.P. Armstrong, R.P. Ayotte.

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## 1. INTRODUCTION

In 1979 a study (1) was carried out to review the application of advanced mining technology to deep underground mines. This study identified areas where advanced technology could improve productivity, provide a safer working environment and lower production costs. In 1980 a subsequent investigation (2) examined the design requirements for the implementation of modern mining concepts under practical conditions within the Onaping No. 2 orebody.

This orebody offers a number of technical challenges to the mining engineer. The orebody is tabular, flat dipping, and lies below 1,000 metres within a difficult geological setting. The concepts established in the 1980 report were:

i) to develop mining methods which are inherently safer than conventional cut-and-fill mining

ii) to be capable of producing 500 tonnes per shift from a single stope

iii) to produce optimum economic results

iv) the latest technology in mine equipment and material handling systems should be utilized

The principal recommendation of the 1980 report was that a small scale bulk mining trial should be undertaken in an area which allowed the application of these concepts. The present project is a continuation of previous studies established between Falconbridge Limited and CANMET concerning the feasibility of implementing bulk

mining practices at depth. The smaller Onaping No. 3 orebody was chosen as the site for the bulk mining trial because:

i) The area is geologically and geometrically similar to the No. 2 orezone.

ii) The area could be developed and in production to assess the proposed mining method prior to the scheduled production from the No. 2 orezone.

The principal conclusions derived from the No. 2 orezone feasibility which pertain to the present study are:

i) The mining method selected for primary extraction is longitudinal blasthole stoping

ii) Safety and environmental aspects are of prime consideration. Inroads towards this objective will be achieved by:

- more automation and mechanisation
- design of stable and well supported excavations
- application of artifical support and ground monitoring programs

iii) It is proposed to prove the technical and economic viability of the following:

a) inclined longitudinal blasthole mining

b) the stability of an artificially reinforced inclined hangingwall

c) blast design guidelines to achieve good fragmentation and controlled muckthrow

d) the maximum utilisation of development

## 1.1 Objective of the Project

It is important to note that the proposed "Bulk Mining Design for Onaping No. 2 Orebody" contained significant design considerations which departed from past practice to ultimately provide a more economic mining method. The original concept includes the objective of high production through the use of stope conveyors feeding directly into the ore-pass system. No intermediate crusher was planned. Good primary fragmentation from blasting under unfavourable geological conditions is essential to eliminate secondary handling.

The No. 3 zone trial study is a test of the mining method only. However, the considerations contained within the feasibility report apply to the present study.

The initial rock mechanics approach towards the bulk mining trial is highly conservative. The objective is to implement inclined longitudinal blasthole stoping within the Onaping No. 3 orebody. If the objective is achieved and the concept proved viable, then the economics of the method will be optimised.

## 1.2 Objective of the Report

On May 21, 1982 Falconbridge Limited announced the closure of Onaping Mine for an indefinite period following the summer production shutdown. To date the closure is still in effect with no plan towards resuming operations within the immediate future.

This has led to amendments to the original contract which pertains to this study. The objective of this report is fulfillment

of the amended contract by presenting the mining concept and providing a record of the project progress to date.

Should Onaping Mine resume operations, the current project wouldcontinue at the discretion of Falconbridge Limited and CANMET.

## 2. STATUS OF THE PROJECT

### 2.1 Progress to date

At the time of the Onaping mine closure the No. 3 Zone bulk mining trial had progressed to the stage of the first panel slot blast. All the development, hangingwall support and instrumentation as proposed was completed for mining the first panel. A raise had been driven from the 3700 level undercut to the first sublevel. The slot area and three lines of production blastholes had been drilled. Access development to the second sublevel was underway when the closure of the mine prevented further progress on the project. Any instrumentation which could deteriorate was salvaged and will be replaced when operations resume.

## 3. BACKGROUND

### 3.1 Mining History

The Onaping No. 3 orezone is a low dipping, low grade orebody with a weak norite hangingwall which was laid out for cut-and-fill mining.

Initial production mining of the No. 3 Zone began in 1979 when a slusher cut-and-fill stope, 38-1360, was started 6 metres above the

3875 level and an exploration stope, 38-1370, was started on the 3700 level. The exploration stope was ramped from the 3700 level down into the ore to intersect the hangingwall. Following the hangingwall contact, the stope was drifted to the east and west to define the ore limits with the plan of developing a large post pillar or rib pillar stope.

The remainder of the orebody from 3700 level to 3050 level required fill-in diamond drilling which was undertaken until mid-1981 when it was discontinued. The diamond drill program made only minor changes to the ore outline, in particular a hangingwall ore stringer.

Both stopes started in 1979 exceeded the diamond drill core grades and more ore than originally projected was mined. The 38-1360 stope was still mining in June, 1982 and is now 13.7 metres above the 3800 level. The 38-1370 stope was closed down on January 1, 1981.

### 3.2 Site Selection and Location

The primary site selection criteria were:

a) the area is geologically and geometrically similar to the No. 2 orezone.

b) the area can be developed and producing so that the mining method can be studied before overcut development begins in the No. 2 orezone.

All areas of the No. 3 orezone were considered as possible site locations. All but one area was eliminated due to excessive development requirement and interference with the mining of the overall block. The area chosen was a small ore block at the extreme

east section of the main zone. This section of ore lies between the 3700 level and the 3050 levels and is separated from the main ore zone by a low grade block (Figure 2).

Only the ore between the 3700 level and 3550 level was developed as the trial stope area. This site was considered the best location because it does not interfere with the two existing cut-and-fill stopes, and contains sufficient tonnage to provide a realistic test of the mining method.

The trial stope location required a minimum amount of development and should the mining method prove unsuccessful, the trial stope ore could be abandoned and the development utilised in mining the larger west section of the orezone. Should the mining method prove successful, the development could be extended to mine the West block from the 3700 level to the 3225 level, employing the same basic mining method (Figure 3).

### 3.3 Mining Method

The primary mining method selected for the No. 2 orezone was longitudinal blasthole stoping. This method involves footwall access drifts with an overcut and undercut at each 12 metre vertical height of ore. Cablebolts were to be installed as the primary ground support with 114 mm diameter blastholes drilled from the overcut to intersect the undercut. Mining could begin at either end of the ore blasting into a slot raise. After each blast the ore would be removed from the undercut by a remote control scooptram. Once all the broken ore had been removed waste would be dumped into the void from the overcut to support the hangingwall.

The longitudinal blasthole stoping method as applied to the No. 3 orezone differs from the original stoping method recommended for the No. 2 orezone. The differences are outlined below:

a) Cablebolt from a Hangingwall Drift

The concept of cablebolting a hangingwall from an undercut has produced several areas of concern:

i) the length of the cablebolt holes is increased since the holes must pass through the ore to intersect the hangingwall.

ii) installing and grouting cablebolts at some point within the hole rather than at the collar would be difficult and time consuming (Figure 4).

iii) monitoring the effect of cablebolts and blasting on the hangingwall would be difficult using conventional instrumentation without a drift in the hangingwall.

iv) cablebolting from an overcut or undercut would extend the length of time required for the development of each block.

v) installing cables in down-holes is faster and easier.

vi) cablebolting from a hangingwall drift creates a progressively tighter bolt pattern into the hangingwall which should halt the progression of caving into the

hangingwall. Cablebolting from the undercut creates the opposite condition (Figure 5).

b) Access Drift in the Hangingwall

The trial stope block is of low grade and low tonnage, and is not able to handle the cost of a footwall access drift and hangingwall cablebolt drift. Therefore the footwall access drift was eliminated and a hangingwall drift planned for cablebolting and access for monitoring.

Some advantages of the hangingwall drift are:

i) a decrease in total drift development.

ii) cablebolts could be installed as the overcut is being excavated.

iii) movement of the hangingwall can be continually and accurately monitored.

Some concern was expressed regarding the hangingwall drift as a permanent access with the possibility of slower development rates due to blocky ground (common in the dark norites), and the resulting need for extra support. It was considered that controlled blasting, perimeter blasting, and resin grouted rebars installed ahead of the round would control this problem. Resin grouted rebars were intended to be used as permanent support in this drift.

Concern was expressed that ground problems might be encountered due to undercutting of the hangingwall drift with the main stope. Cablebolting the intervening hangingwall on a 1.8 x 1.8 m pattern should provide sufficient stability for the duration of mining the panel.

The hangingwall drift was not planned as a permanent drift and on completion of mining in each panel the hangingwall drift would be back filled (Figure 6). In the trial stope the hangingwall drifts will not be filled so that monitoring of the lower panel can be done during blasting of the second and third panels.

### 3.4 Development Systems

The trial stope has a vertical height of 45 metres from the 3700 level drift to the 3550 level base of rail. The block is divided into three 12 metre high panels. Access to each mining horizon is via a service raise for intermediate sublevels and from the 3700 and 3550 drifts at the level elevations.

Ore and waste are passed through an existing raise system connecting the first sublevel. Ore from the 3700 level undercut is passed through a new raise down to the 3800 level.

The 3800 level haulage presents a potential bottleneck with a current peak tramming capacity of 7300 tonnes per month. It is incapable of handling both ore from the 38-1360 cut-and-fill stope and

peak production from the trial stope. To improve production rates it would be necessary to tram No. 3 zone ore on both the 3700 and 3800 levels.

The development and production plans for the first panel are as follows: (Figure 7)

a) The 3700 level undercut will be developed using a JS 220 scooptram.

b) The 37+1 sublevel hangingwall drift will be developed with a JS220 scooptram as a priority heading to allow cable bolting to start as soon as possible. The ore overcut will be completed and blasthole drilling undertaken using a Tamrock drill. The scooptram will remain on this level to move waste into the blasted stope.

c) The scooptram from the 3700 level will be moved to the 37+2 sublevel and a JS500 scooptram with a remote control will be assembled on the 3700 level.

d) A slot raise will be driven and blasting of the first panel will start.

e) Waste from the 37+2 development drift will be used as backfill in the first panel once the JS500 has removed the ore. Provisions have been made to tap the 3550 level waste pass for additional backfill. This waste will be trammed along the 3550 level, dumped into the 37+1 sublevel, or later the 37+2 sublevel, and hauled by JS220 to backfill the stope.

f) Once the first panel has been mined it will be filled with 32:1 cemented tailings fill and waste rock. During this time the second panel will be developed for mining.

The overcut has been excavated to a width of 14 metres over the full length of the orezone (Figure 8). The original proposal was to open a 12 metre wide stope but a wider overcut was required to accommodate the footwall hole drilling due to the dip of the orebody.

The undercut will remain at a 12 metre width as no large hole drilling will be done from here. The ore below the undercut has been abandoned and no mat is planned. Minimal ore will be lost due to the dip of the orebody, and any ore to the south can be mined at a future date by the 38-1360 stope (Figure 10).

The overcut hangingwall drift has been driven 15 metres from the orezone in plan giving a minimum distance from the orezone hangingwall of 10.6 metres. Cables and instrumentation will be installed from this drift and the effect of blast vibration and the open stope on this drift monitored.

The slot raise in the first panel is at the east end of the orezone. Undercut mucking will start from the east and retreat to the west.

## TABLE 1 ONAPING MINE #3 ZONE SCHEDULED DEVELOPMENT

| DEVELOPMENT | YEAR | UNITS m |
|---|---|---|
| 3700 Service Raise Dr. | 1981 | 30 |
| 3500 Service Raise Dr. | 1981 | 24 |
| 3700 Service Raise | 1981 | 54 |
| 3700 + 1 O.T.R. | 1981 | 14 |
| 3700 + 1 Overcut Dr. | 1981 | 183 |
| 3700 Cable Bolting | 1981 | 3,731 |
| 3700 Undercut Slash | 1981 | 583 $m^3$ |
| 3700 + 2 Overcut Access | 1981 | 15 |
| 3700 + 2 Overcut Dr. | 1982 | 168 |
| 3700 + 2 O.T.R. | 1982 | 14 |
| 3700 + 2 Cable Bolting | 1982 | 3,731 |
| 3550 O.T.R. | 1982 | 14 |
| 3550 Overcut Drift | 1982 | 50 |
| 3550 Overcut Drift | 1983 | 117 |
| 3550 Cable Bolting | 1983 | 3,731 |

## TABLE 2 ONAPING MINE #3 ZONE TRIAL STOPE SCHEDULED PRODUCTION

| PRODUCTION | TONNES | YEAR |
|---|---|---|
| 3700L Undercut | 9,072 | 1981 |
| 3700 + 1 Overcut | 9,072 | 1981 |
| TOTAL 1981 | 18,144 | |
| 3700 + 1 Panel | 36,289 | 1982 |
| 3700 + 2 Overcut | 9,072 | 1982 |
| 3700 + 2 Panel | 18,144 | 1982 |
| TOTAL 1982 | 63,503 | |
| 3700 + 2 Panel | 18,144 | 1983 |
| 3550L Overcut | 9,072 | 1983 |
| 3550 Panel | 36,289 | 1983 |
| TOTAL 1983 | 63,503 | |

### 3.5 Production Schedule

Ore production is restricted by the capacity of the ore haulage tram system on the 3875 level. Ore production could be improved by transferring some ore to the 3700 level but it was decided to accept a reduced rate to allow for experimentation within the trial stope block. Stope development and the Production rates per year are shown in Table 1 and 2.

## 4. GEOLOGY

### 4.1 General

The Onaping ore deposit is one of a family of several known orebodies which occur at or near the base of the Nickel Irruptive in the northwest corner of the Sudbury basin. These deposits are restricted to an intensely mineralised length of the Irruptive contact located in the Levack-Onaping area. The geological setting in the Onaping area is typical of the northern rim of the Sudbury basin, dominated by the Nickel Irruptive which has been intruded between the Interior Basin sedimentary and volcanic rocks to the south and the footwall Granitic Complex to the north. All the consolidated rocks in this area are Precambrian in age (Table 3).

Onaping mine is located in Levack Township, with access by private road from the Sudbury - Levack highway. The orebody is comprised of disseminated sulphides which occur within the nose of a broad fold of Dark Norite and Dark Norite Breccia. The hangingwall

is composed of Felsic Norites and the footwall rocktype consists of Feldspathic Gneiss.

Onaping is located in plan between the Onaping No. 1 and No. 2 Zones to the west and Craig Mine to the east. Laterally the economic limits of the orezone fall between 13400E and 14100E (Hardy Mine grid) with vertical limits between the 3375 and 3875 levels.

The section of ore proposed for the trial stope lies between the 3700 level and the 3550 level and is separated from the larger west block by a 30 metre waste block.

### 4.2 Felsic Norite

The Felsic Norite grades from an overlying transition zone, the exact contact of which is often difficult to determine in the field. Felsic Norite is a grey, medium to coarse grained, equigranular, massive rock, composed of plagioclase, potash feldspar, quartz and biotite. Pyroxenes are usually altered. Minor accessories include magnetite, ilmenite, sulphides and apatite. It is usually strongly altered and is quite distinctive in the field. Felsic Norite is not an important host rock for sulphides.

### 4.3 Dark Norite

The Dark Norite is a lithological and mapable unit which forms discontinuous bodies underlying the Felsic Norite. The contact with the Felsic Norite can be either sharp or gradational over a few metres.

TABLE 3 - TABLE OF FORMATIONS

| | | |
|---|---|---|
| LATE PRECAMBRIAN<br>Keweenawan | Younger Diabases | Olivine diabase<br>Quartz diabase |
| MIDDLE PRECAMBRIAN<br>Huronian | Late Acid Intrusives | Aplite, epidote, and siliceous dikes |
| | Post-irruptive Brecciation | Late Granite Breccia....Granite Breccia, Contact Breccia |
| | Nickel Irruptive | Micropegmatite<br>Transition zone rocks<br>Norite ..... Dark Norite ..... Hanging-wall Breccia, Basic Norite, Mafic Norite<br>Norite ..... Felsic Norite |
| | Whitewater Group | Chelmsford 'sandstone"<br>Onwatin slate<br>Onaping volcanics |
| | Pre-irruptive Brecciation | Levack breccia |
| | Early Acid Intrusives | Pegmatites and aplites |
| | Older Diabase | Quartz diabase dikes |
| EARLY PRECAMBRIAN<br>Pre-Huronian | Granitic Gneiss Complex | Tonalitic gneiss, Mafic Gneiss<br>Early volcanics and sediments? |

The Dark Norite is a dark grey, coarse to fine grained, equigranul massive rock composed of biotite, hornblende, quartz, apatite, ilmenite and magnetite. Small amounts of disseminated sulphides are often present. The percentage of alteration varies, but in general the Dark Norite has a fresher appearance than the Felsic Norite. According to textural features, the Dark Norite is subdivided into several varieties:

1. Unbrecciated Dark Norite sometimes grades or sharply converts into a brecciated noritic rock-Dark Norite Breccia. Dark Norite Breccia is a dark grey, fine to medium grained, massive, inequigranular rock composed of a Dark Norite matrix and a variety of xenoliths and xenocrysts. When the percentage of inclusions reaches the arbitrary figure of 30%, the term Hangingwall Breccia is used. The noritic matrix has a fresh and not always typically igneous appearance. The matrix is composed of labradorite, hypersthene, augite with minor biotite, magnetite and quartz. Sulphides are finely disseminated and often form blebs or irregular stringers in the matrix. Economic amounts of sulphides are sometimes present in this rocktype. Xenoliths represent inclusions of ultramafic and mafic rocks of a noritic parentage. The origin of these inclusions remains uncertain, but it is thought they are of footwall origin.

2. Hangingwall Breccia differs from the Dark Norite Breccia only in the amount of inclusions. When the percentage of inclusions exceeds the figure of 30% the term Hangingwall Breccia is used. The Hangingwall Breccia usually occupies the basal portion of the noritic rocks. This rock may become gneissic and more feldspathic close to the footwall. This more feldspathic type occurring at the contact with footwall formations is sometimes called the Contact Breccia and represents a transition to the Late Granite Breccia. The Hangingwall Breccia is an important host rock for sulphides along the north range.

3. Zones of Late Granite Breccia sometimes occur between the bottom of the Nickel Irruptive and the footwall gneiss complex. These zones generally follow the Norite contact and tongues of the Granite Breccia often extend into the Hangingwall Breccia. The contact with the Hangingwall Breccia is usually gradational through an intermediate stage known as the Contact Breccia. The contact with the footwall gneiss is also gradational, though variable. The Late Granite Breccia is a very important host rock for sulphides and may be divided into two sub-units based upon the colour of its appearance in the field.

## 4.4 Feldspathic Gneiss

The Feldspathic Gneiss or Granite Gneiss is found only within the north range footwall complex of the Sudbury Basin. The terms Feldspathic and Granite Gneiss are synonymous and are interchangeable in Falconbridge usage.

Feldspathic Gneiss is a pinkish to light grey, fine to medium grained rock with a gneissosioty defined by irregular sub-parallel mafic bands. The bands are composed of aggregates of augite, hornblende, biotite and secondary minerals set in a fine grained matrix of andesine and quartz. The Feldspathic Gneiss contains numerous mafic inclusions which probably represent fine grained mafic rocks. The Feldspathic Gneiss grades into Mafic Gneiss or Granite Breccias and into underlying massive or weakly foliated granitic or quartz dioritic rocks.

1. When the percentage of mafic minerals in the Feldspathic Gneiss exceeds 50% the term Mafic Gneiss is arbitrarily used. These two varieties often grade into each other which can cause difficulty in underground mapping.

The sulphide mineralisation varies from irregular, weakly to heavily disseminated pyrrhotite-pentlandite patches to massive bands and stringers generally striking east-west and dipping 40° to 50° south. The strike length of the orebody varies from 60 to 75 metres and the thickness from 6 to 18 metres. Locally small pockets of

magnetite and epidote mineralisation high in pentlandite occur adjacent to the footwall. Laboratory results on samples submitted in 1981 indicated the magnetite would not cause any milling problems.

## 4.5 Rock Mass Structure

Site specific structural information was obtained by a process of ongoing level mapping as the access development for Onaping No. 3 Zone progressed. These areas include:

3707 Drift, 3702 Drift, 37-0 Sub-Level Access Drift, 37-1380 Stope 37-0 Sub-Level, 37-1398 Cross-Cut, 3705 Drift, 37-1 Service Drift, 37-1 Hangingwall Drift, 37-1380 Stope 37-1 Sub-Level.

Data taken from the geological mapping are presented as lower hemisphere contoured polar projections for each lithological unit encountered during development stages, (Figures 11 to 16). Jointing within the norites was compared with the results of level mapping in the 2050 and 3050 East Exploration Drifts (Figure 17) to establish consistency and define the number of joint sets observed. The attitude of the average joint sets were found as follows:

| | DIP | DIP DIRECTION* |
|---|---|---|
| Joint Set I | 80° | 280° |
| Joint Set II | 55-85° | 90° |
| Joint Set III | 45° | 340-025° |
| Joint Set IV | 30-40° | 180°+ |

* Dip Direction is with respect to mine north (Hardy Mine Coordinates).

This information compares well with the findings of the No. 2 Zone investigation.

Set IV is not shown in Figs 12 and 19 and was not picked up during structural mapping. This set parallels the norite/gneiss contact and has been identified at numerous other mines in the vicinity. The spacing of these sets ranges from 0.25 to 1.5 metres. The joints are commonly planar, semi-continuous and tight, though slickensiding is not uncommon.

Jointing within the footwall gneiss is less prominent (Fig. 11) with two main sets identified as follows:

| | DIP | DIP DIRECTION |
|---|---|---|
| Joint Set I | 70° | 344° |
| Joint Set II | 40-45° | 319° |

Joint spacing ranges from 1 to 3 metres, with occasional frequencies of 3-4 joints/metre occurring locally. Typically the footwall jointing is planar, continuous and tight with a higher order of surface roughness than in the norites, slickensiding is not uncommon.

A geometrical analysis of intersecting planes for the norites is shown in Fig. 19, a range of friction angles have been used:

50° represents the peak intact rock strength value.

30° represents the adjusted rock mass friction value which accounts for structural influences. It can be determined that:

i) No significant wedges are present in the overcut or undercut backs.

ii) Potential instabilities occur by intersecting planes and failure occurring by sliding to the north along joint set III.

iii) Joint sets I and II can be assumed to be the same joint set with respect to the degree of instability and form the role of release surfaces.

iv) The principle direction of sliding daylights within the hangingwall and lies between the rock mass and intact rock friction angles.

v) The footwall of the orezone is structurally stable.

The prominent failure mode of the hangingwall is determined to be the sliding and unravelling of small blocks which will undercut the up-dip hangingwall. The progression of minor sloughing to uncontrolled caving is a feature to be accounted within the support design.

Faulting has been observed within the No. 3 Zone orebody, the distributions of which are shown in Figure 18. Commonly the faults are planar, tight, with minor gouge, and dip steeply to the northwest. Traces observed within the 37-1 Sub Level overcut and the 37-0 undercut suggest good continuity. The faults occur both individually and as conjugate shears, evident within the

conjugate shears, evident within the rock mass by the presence of occasional minor water seepage after being intersected by development. Displacements are unknown, though thought to be minor, in one instance slickensiding and sympathetic shearing have produced an observable shear zone up to 1 metre wide.

### 4.7 Groundwater

Problems associated with groundwater are uncommon at Onaping mine, the ground in the area of No. 3 Zone is dry. For the purpose of the analyses the presence of groundwater has been omitted and considered to perform a negligible role within the rock mass.

## 5. ROCK PROPERTIES

### 5.1 Mechanical Properties

The modulus of deformation and Poisson's ratio were determined in the field as part of an in situ stress measurement program. The 150 mm overcored rock samples with strain cell were tested in biaxial compression. The response of the nine gauges of the strain cell were monitored during loading and unloading in the 0-20 MPa range.

Detailed results are presented in Appendix 1 to the 1980 report (2). The final values of elastic parameters were determined as follows:

| | Deformation Modulus (GPa) | | Poisson's Ratio | |
|---|---|---|---|---|
| Rock Type | Loading | Unloading | Loading | Unloading |
| Felsic Norites (Hangingwall) | 58 | 59 | 0.18 | 0.19 |
| Dark Norites (hangingwall) | 84 | 82 | 0.32 | 0.29 |
| Mafic Gneiss (footwall) | 78 | 74 | 0.27 | 0.27 |

All available data on rock strengths from Falconbridge properties with similar geological environments to the Onaping No. 3 orebody was reviewed and analysed. Additional point load tests on EX core from the Onaping No. 2 orebody were conducted to verify the uniaxial compressive strengths. The intact laboratory strengths are summarised below:

**TABLE 4 - ONAPING MINE ROCK STRENGTH DATA** (2)

| Rock Type | Uniaxial Compressive Strength; Co (MPa) | Tensile Strength; To (MPa) | Cohesion So (MPa) | Friction Angle; (degrees) |
|---|---|---|---|---|
| Felsic Norites (Hanging wall) | 237 | 16 | 24 | 53 |
| Dark Norites (Hanging wall) | 240 | 15 | 23 | 51 |
| Late Granite Breccia (orebody) | 279 | 26 | 40 | 54 |
| Footwall Mafic Gneiss | 331 | 25 | 42 | 55 |
| Footwall Felds-pathic Gneiss | 363 | 19 | 37 | 55 |
| The ratios Co/So and Co/To are fairly consistent for the five rock types. Co is 9 times the cohesive strength and 15 times the tensile strength of intact rock. | | | | |

## 5.2 Uniaxial Compressive Testing

Twenty-one samples of AX diameter core were tested at the University of Toronto to obtain values for the elastic constants and uniaxial compressive strengths of the Onaping No. 3 Zone rocktypes. The dimensions and identification of the samples and the results of the uniaxial compression tests are summarised in Appendix B. The rocks can be divided into two groups based upon their stress-strain behaviour:

a) The Feldspathic and Mafic Gneiss

The complete stress-strain behaviour of these samples could not be obtained with the testing system used. The capacity of the rocks to resist load under post-peak stress conditions decreased faster than the load applied by the machine could decrease. Consequently the samples failed explosively at the peak load.

Samples in this group have considerably higher values of compressive strength, $C_o$ and elastic modulus (E), than the other group described below. The low values of $C_o$ for samples 3-F-1A and 5-L-1A are the result of failure along pre-existing flaws.

b) Dark Norites and Mineralised Dark Norites

For these samples the complete stress-strain behaviour was obtained. A study of the load-displacement curves show that the samples sustained 60-70 per cent of their peak strength in the post peak stress state.

The samples in this group have considerably lower values of compressive strength than the Gneiss samples.

The Modulus of Resilience represents the capacity of a material to store strain energy per unit volume to the elastic limit.

From Table 9 it can be seen that the Felsic Gneiss of the Onaping No. 3 Zone footwall has a higher modulus of resilience and potential towards rockbursting than the Dark Norites.

The uniaxial test results produced lower Poisson's Ratio values than previously determined for Onaping rocktypes. When asked, the University of Toronto failed to provide a satisfactory explanation concerning these results, it has been assumed that an error was made in their calculation and the values discounted.

## 5.3 Triaxial Compressive and Tensile Testing

Twenty samples of BX diameter core were tested at Queen's University to obtain values of the triaxial compressive and tensile strengths. Of the five designated rocktypes, three samples per rocktype were tested in triaxial compression (at confining pressures of 6.9, 13.8 and 20.7 MPa respectively) and one sample per rocktype was tested in tension by direct pull. The results of the tests are summarized in Tables 10 to 12.

## 5.4 Ground Conditions

Initial indications concerning the likely ground conditions within the No. 3 zone were based upon experiences encountered under

similar circumstances and appeared unfavourable. As the project progressed, and site specific data was obtained, the understanding of the No. 3 zone ground conditions improved.

Previous experience in cut-and-fill mining in the Onaping norites has encountered structural failure to a height of 3 metres when a 12 metre mining span has been exceeded. This limit was applied to the No. 3 zone stope width during the planning stages of this project. The orebody favours this reduction due to ore offshoots, patches of low grade and the disseminated nature of the norite mineralisation.

The No. 2 zone feasibility report indicated that generally poor ground conditions could be encountered in the immediate hangingwall norites. Pressured grout injection was recommended to improve local ground conditions and a grouting trial was conducted in advance of ventilation raise construction from the 3100 level. The trial was unsuccessful with little or no grout penetration into the norites, indicating that ground conditions can be variable.

A similar improvement over the expected ground conditions was found during the development of the 3700 level undercut, first sublevel overcut and hangingwall drift.

The variability of the ground conditions was observed during hangingwall cablebolt grouting at the extreme western stope limits. Cement migrated into adjacent boreholes suggesting the presence of an open fracture pattern within the Dark Norite Breccias.

RQD's from the borehole investigation of No. 3 zone are shown in Appendix C. It was intended to correlate estimated RQD's from core photographs to actual RQD's from borehole core and in situ RQD's from borehole camera observations. Equipment failure has prevented the use of the borehole camera to date. The estimated RQD's from core photographs are consistently lower than those calculated from borehole core. A discussion of the RQD results is presented in the following section on rock mass classification.

Prior to the closure of Onaping mine, ground problems associated with rockbursting were encountered in driving the footwall access ramp to No. 2 zone below the 4000 level. To date there has been no evidence of similar conditions within the Feldspathic Gneiss of No. 3 zone.

## 5.5 Rock Mass Classification

The design of a numerical model requires a division of the geological environment into units which exhibit similar rock mass properties and behaviour. During the preliminary stages of a study the available data is often approximate and qualitative and a rock mechanics classification provides a useful approach to quantify this data in broad terms.

Data collection has proceeded throughout the duration of the project and is continually revised. The primary sources of information include:

a) historical mine records

b) previous experience of Onaping ground conditions

c) diamond drilling records

d) borehole logs

e) test results of rock properties

f) in situ stress determination

g) geological level mapping

Previous sections of this report describe the geology, structure and rock properties of the No. 3 zone. Appendices C and D present the results of borehole core logging, RQD's, and rock mass classification according to the system proposed by Beiniawski, 1974. The range of RQD varies from 22-96%. The range of CSIRO rating varies from 43-83. All the rocktypes encountered fall within the fair to good rock groups of Beiniawski and of Barton (Q = 25 - 46).

The average values of each rocktype can be summarised as follows:

**Table 5 - Summary of Geomechanical Groups - Onaping No. 3 Zone**

| Rocktype | Fractures Per Metre | RQD | CSIRO Classification | | |
|---|---|---|---|---|---|
| | | | Rating | Class No. | Description |
| Feldspathic Gneiss (FGN) | 3 | 68.9 | 83 | II | Good |
| Late Granite Breccia (LGBX) | 4-6 | 63.3 | 74 | II-III | Fair to good |
| Dark Norite Breccia (DNBX) | 4-5 | 55.8 | 73 | II-III | Fair to good |
| Dark Norite (DNOR) | 3 | 63.4 | 81 | II | Good |

Locally poor ground, RQD< 40%, CSIRO rating 25-40 occurs within the Late Granite and Dark Norite Breccias. Historically, within Falconbridge experience, ground instability has been associated with an RQD of less than 40%.

The in situ rock mass properties and strengths of the geomechanics groups were determined by Deere, et al (1968) from the relationship between in situ rock stiffness and fracture frequency. Experience at Falconbridge has shown this correlation provides a reasonable estimate of in situ rock mass strength. Accordingly the laboratory intact properties have been adjusted for input into the numerical analysis (Table 7).

It should be noted that the geomechanics groups are based upon subjective data and should be treated with caution. The adjusted rock mass properties are considered conservative and represent the worst expected ground conditions.

## 5.6 In Situ Stress

One of the factors which affect the successful application of numerical models for excavation design is the reliability with which the in situ (pre-mining) stress field can be predicted.

Rock stresses increase with depth. While the vertical stress component generally increases linearly with depth and correlates well with the overburden pressure, the horizontal stress components are influenced by the geological and tectonic environment and are

difficult to predict. A reliable estimate of the in situ stress field can, therefore, be made only from field measurements.

Overcoring to determine in situ stress levels has been conducted at Onaping Mine (2). Three sites were located in the hangingwall andhangingwall and footwall rocks, approximately 200 metres apart in depth, on the 4050, 4700 and 5500 levels. The results are presented in terms of the principal stress magnitudes and orientations in Table 6.

**TABLE 6**

**PRINCIPAL STRESSES AND THEIR ORIENTATIONS AT ONAPING MINE**

| | MAJOR PRINCIPAL STRESS 1 | | INTERMEDIATE PRINCIPAL STRESS 3 | | MINOR PRINCIPAL STRESS 3 | | | |
|---|---|---|---|---|---|---|---|---|
| | MAGN. (MPa) | ORIENT.* | MAGN. (MPa) | ORIENT. | MAGN. (MPa) | ORIENT | $\sigma 2 : \sigma 3$ $\sigma 1 : \sigma 1$ | $2(\sigma 1)$ $(\sigma 2 + \sigma 3$ |
| 4050L | 64.50 | 25/318 | 39.27 | 32/064 | 32.68 | 48/200 | 0.61:0.51 | 1.80 |
| 4700L | 84.48 | 09/293 | 50.32 | 12/027 | 40.71 | 72/175 | 0.59:0.59 | 1.86 |
| 5500L | 81.55 | 09/276 | 48.30 | 27/013 | 35.42 | 62/168 | 0.59:0.43 | 1.95 |

* Shown in terms of dip (positive down from horizontal) and bearing clock-wise from mine north.

## 6. NUMERICAL ANALYSIS

Part of the geotechnical investigation of the No. 3 orezone consisted of the numerical analysis of the proposed mining method. Two-dimensional plan and transverse finite element and three-dimensional MINTAB models were analysed. The objectives of the analyses were twofold:

i) to form a basis for ground control guidelines

ii) to demonstrate the accuracy of rock mass classification and the resulting rock strengths as input for modelling through comparison to the results of in situ instrumentation

### 6.1 Model Selection

The static two-dimensional finite element model was used in this study because of its capability of simulating a greater variety of rock types and mining geometries than alternative methods available. The two-dimensional analysis allows the most economic overall evaluation of mine stability.

A number of assumptions apply to the finite element models used. These are:

1. The materials modelled are considered to be homogenous, time independent and to follow a linearly elastic and isotropic behaviour.
2. A plane strain condition exists.
3. A constant in-situ stress field is assumed.

4. The failure criteria are based upon maximum tension and Mohr-Coulomb theories. The modes of failure are tension, compression and shear.
5. Mine openings are instantly excavated or filled.

MINTAB is a computer program developed for analysing an extraction sequence in a tabular orebody which is flat-dipping at any angle up to 90° to the horizontal(8). The program analysis is three-dimensional and is based upon the displacement discontinuity technique.

A basic assumption is that the mine workings are sufficiently deep so that the influence of the surface upon stresses and displacements near the mine workings can be neglected.

The capabilities of MINTAB are:

1. Elastic behaviour of orebody and surrounding rocks has to be assumed.
2. The region outside the mine grid is considered to be incompressible.
3. Pillar yielding or fracturing is not simulated and support systems such as rock bolts or backfill cannot be modelled with the present version.
4. The mine should be sufficiently deep so that the ground surface will not influence the results.

## 6.2 Finite Element Models

Two finite element models were constructed to analyse the stress conditions within the rock mass. The input values for material properties represent adjusted intact laboratory properties to reflect the rock mass condition (Table 7). The degree of reduction represents a combination of factors including the number and quality of test results, the rock structure, and experience gained through the back analysis of previous studies within the Sudbury Basin.

Measured Stress Values (MPa):

| | | | |
|---|---|---|---|
| $\sigma$ vert | = 31.6 | $\sigma$ e-w | = 54.4 |
| $\sigma$ horiz | = 45.9 | $\sigma$ n-s | = 42.3 |

Values for tailings fill and cemented rockfill have been taken from the CANMET report "A Computer Model to Predict the Free-standing Height of Backfill in Pillar Recovery Operations" by Dames and Moore, March 1977.

### 6.2.1 Plan Model

Located at a depth equivalent to the mid-height of the second panel, (1116 metre elevation), the point of the maximum theoretical hangingwall deflection. This model examines:

i) hangingwall stress levels and displacements

ii) effect of an extraction sequence

iii) the location of access openings

TABLE 7 SUMMARY OF ROCK STRENGTH REDUCTIONS FOR INPUT TO NUMERICAL MODELLING

| Rocktype | Test Source | Density Kg/m$^3$ | Uniaxial Strength(MPa) | Tensile Strength(MPa) | Cohesion MPa | Young's Modulus MPa | Poisson's Ratio | Friction Angle |
|---|---|---|---|---|---|---|---|---|
| FGN | U. of T.<br>Queens U.<br>Reduction | | 141.3<br>x 0.7 | 9.6<br>x 0.7 | 29.0<br>x 0.7 | 58,605<br>x 0.9 | 0.28 | 44 |
| | | 2704 | 84.7 | 6.7 | 20.3 | 52,744 | 0.28 | 44 |
| LEBX | U. of T.<br>Queens U.<br>Reduction | | 131.3<br>x 0.5 | 9.4<br>x 0.5 | 24.1<br>x 0.5 | 49,642<br>x 0.9 | 0.26 | 49 |
| | | 3392 | 65.6 | 4.7 | 12.1 | 44,672 | 0.26 | 49 |
| DNBX | U. of T.<br>Queens U.<br>Queens U.<br>Reduction | | 81.2<br>x 0.6 | (5.6)<br>7.2<br>x 0.6 | (23.4)<br>15.9<br>x 0.6 | 53,089<br>28,958<br>x 0.9 | (0.23)<br>0.25 | (51)<br>48 |
| | | 3200 | 48.7 | 4.3 | 9.5 | 36,920 | 0.25 | 50 |
| DNOR | U. of T.<br>Queens U.<br>Reduction | | 128.0<br>x 0.7 | 8.7<br>x 0.7 | 57.9<br>x 0.7 | 85,495<br>x 0.9 | 0.30 | 32 |
| | | 2960 | 89.6 | 6.1 | 40.5 | 76,943 | 0.30 | 32 |
| Tailings Fill (30:1) | CANMET | 1890 | 0.1 | 0 | 0.01 | 100 | 0.30 | 30 |
| Cemented Rock Fill | CANMET | 2000 | 5.6 | 0.3 | 0.1 | 2097 | 0.33 | 40 |

A two-dimensional plan model is incapable of simulating the inclined condition of the hangingwall and footwall. A true plane-strain condition does not exist and the results are used only for guideline purposes.

6.2.2 Transverse Model

Taken along section 1400E where the overall geology is considered representative of the whole orebody and the zone of brecciation is at its widest. This model examines:

i) hangingwall stress levels and displacements

ii) the extraction sequence

iii) the location of access opening

iv) the model sensativity to a variation of rock strengths

v) the model sensativity to a variation of K

## 6.3 Finite Element Results

a) Plan Model

Two cases were run to examine the stress distributions around the proposed openings.

Case 1 (Figure 20)

An intact, solid run, with the exception of the 37-1398 Service Raise being open. The geology and stope limits are as shown. The coefficient of tectonic presure (K) is 1.3

Results indicate:

i) high compressive stress occurs within the brecciated norite to the west of the orezone

ii) high compressive stresses and minor potential shear failures occur around the 37-1398 Service Raise

Case 2 (Figure 21)

As in Case 1, with the proposed stope fully open and mined. The results indicate:

i) high compressive stresses occur within the Dark Norite Breccias at the east and west limits of the orezone

ii) potential shear failures develop at both the east and west stope limits to a depth of 1.5 metres

iii) minor tensile zones develop in the hangingwall and footwall to a depth of 1.5 metres

Contours of major principal stress, minor principal stress and maximum shear are shown in Figures 22 to 24.

b) Transverse Model

Seven simulations were examined to determine the stress distributions associated with the proposed mining sequence. An initial intact or solid run provided a virgin ground/pre-mining reference to compare to the induced displacements of later cases.

Case 1 (Figure 25)

The 3700 level undercut, the first sublevel overcut and hangingwall drift, and the access drifts within the area of interest mined and open. The mined span of the overcut is 14 metres, and the undercut 12 metres. The coefficient of tectonic pressure (K) is 1.5 The results indicate:

i) potential tensile failures locally up to 1.2 metres above the back of the undercut and overcut and in the floor of the overcut to a depth of 2.4 metres

ii) high compressive stresses extend 3 metres into the hangingwall and footwall areas of the undercut and overcut

iii) potential shear failure occurs up to 3.6 metres above the undercut back

iv) tensile zones extend 2.4 metres above the undercut and overcut backs and also below the overcut floor.

Contours of major principal stress, minor principal stress and maximum shear are shown in Figures 26 to 28.

Case 2 (Figure 29)

All openings are as in Case 1, in addition the first panel is mined and the second sublevel and hangingwall drift have been excavated. The coefficient of tectonic pressure (K) is 1.5. The results indicate:

i) potential tensile failures up to 1.5 metres above the overcut back, 1.2 metres above the back and 1.5 metres below the floor of the second sublevel

ii) high compressive stresses extend: 6.7 metres into the hangingwall adjacent to the undercut, the maximum predicted stress is 107 MPa; from the footwall of the

first sublevel to the hangingwall of the second sublevel, the maximum predicted stress is 99 and 151 MPa respectively; up to 2.4 metres into the footwall of the second sublevel, the maximum predicted stress is 83 MPa

iii) potential shear failures occur to a depth of 1.5 metres below the second sublevel floor and within the second panel

iv) tensile zones extend up to 6 metres into the hangingwall, 3.6 metres above the overcut and second sublevel backs, 4.8 metres below the second sublevel floor, and over 3 metres into the footwall of the first panel. Minor tensile zones affect the back and floor of the first sublevel hangingwall drift to a depth of 1.2 metres.

Contours of major principal stress, minor principal stress and maximum shear are shown in Figures 30 to 32.

Case 3 (Figure 33)

The first panel is filled with cemented rockfill, the second panel is mined and open. The 3550 level overcut and hangingwall drift are developed and open. The coefficient of tectonic pressure (K) is 1.5. The results indicate:

i) potential tensile failures occur 1.2 metres above the back of the second sublevel and 1.8 metres above the back of the 3550 level overcut. A potential tensile failure extends 3 metres below the floor of the 3550 level overcut.

ii) high compressive stresss extends 7.6 metres into the hangingwall adjacent to the 3700 level undercut, 6 metres above the footwall shoulder of the second sublevel, 3 metres into the hangingwall and footwall of the 3550 level overcut. The respective maximum predicted stresses are: 105, 122, 108 and 95 MPa

iii) potential shear failures occur up to 4.8 metres above the back of the second sublevel, 3.6 metres below the floor of the 3550 level overcut, and 1.8 metres into the hangingwall shoulder of the 3550 level overcut

iv) tensile zones extend up to 12 metres into the hangingwall and 4.5 metres into the footwall of the second panel. The back of the second sublevel, the sidewall of the second sublevel hangingwall drift, the back and floor of the 3550 level overcut are all affected by tensile zones up to 3 metres depth. Contours of major principal stress, minor principal stress and maximum shear are shown in figures 34 to 36.

Case 4 (Figure 37)

The first and second panels are filled with cemented rockfill, the third panel is mined and open. The coefficient of tectonic pressure (K) is 1.5. The results indicate:

i) high compressive stresses occur upto 9 metres into the hangingwall adjacent to the 3700 level undercut, and 3.6 metres above the footwall shoulder of the 3550 level overcut. The maximum predicted stresses are 105 and 132 MPa respectively

ii) tensile zones extend 7.2 metres into the hangingwall and 4.5 metres into the footwall of the third panel. The back of the 3550 level overcut is affected by tension to a depth of 1.8 metres

Contours of major principal stress, minor principal stress and maximum shear are shown in Figures 38 to 40.

c) Sensitivity Analysis of K

To examine the sensitivity of the assigned coefficient of tectonic pressure two cases were simulated using limiting K values of 1.2 and 2.5, (Cases 5 and 6, figures 41 and 42)

Generally, the potential failure zones and stress concentration areas are extended under the influence of the lower K value and diminish with the higher K value.

d) Sensitivity analysis of rock mass strengths

To examine the sensitivity of the assigned rock mass strengths, Case 7 was simulated with 50% values of the previous analyses.

The stress concentrations remain constant, the areas exceeding the rock mass strengths are extended to the limit of the reduced values in accordance with the Mohr-Coulomb failure criteria (Figure 43).

## 6.4 Mintab

As a comparison to Finite Element Analysis the extraction sequence and geometry of No. 3 Zone was simulated using the MINTAB program. The model reef thickness as analysed is 8.53 metres. The primitive stress ratios were calculated as:

CX = 1.34

CY = 1.72

Figure 44 compares the stress levels of coincident points through the hangingwall of the plan and transverse finite element models to the results obtained using MINTAB. It is felt that the comparison falls within acceptable limits, given that:

i) a plane strain condition is simulated within the plan finite element model, which is not the case in practice.

ii) the finite element technique can incorporate a greater number of rocktypes, MINTAB results present a simplified approach.

iii) the nearest MINTAB off-reef point is restricted to a location at a distance greater than the width of a small grid element. Consequently the stresses within the immediate hangingwall limit cannot be defined.

## 6.5 Conclusions of Numerical Modelling

From the results of the numerical modelling the following conclusions can be drawn with respect to the regional and local stability.

i) Mining the No. 3 zone using an inclined blasthole stoping sequence is feasible with the addition of support.

ii) As tensile zones are critical for hangingwall stability, hangingwall support should be installed before mining the adjacent panel. The objective is to improve the overall rock mass cohesion and restrain the development of induced tensile stress.

iii) Artificial reinforcement to a depth of 3 metres is considered adequate to remedy localised potential failures around access openings.

iv) Concentrations of high compressive stress suggest a potential for rockbursting, particularly in the stiffer Feldspathic Gneiss of the footwall. If rockbursting is not encountered during the development stages, the high compressive stress concentrations will produce slabbing and spalling ground. The presence of

compressive stress within the norites will provide confinement in blocky and brecciated ground.

v) Potential shear failures within the mining panels may produce difficulties during the drilling and loading of blastholes. These include stuck drill-strings and off-set and caving boreholes.

vi) The ground above 3550 level and the sill pillar below 3700 level are stable.

vii) Finite element modelling indicates that the introduction of backfill after mining effectively stabilizes the hangingwall.

## 7. SUPPORT DESIGN

### 7.1 Support Design Concepts

The initial concept of support within No. 3 Zone is to provide a conservative design which is successful for mining the first panel. A "design as you go" philosophy will be adopted for the succeeding panels in response to the back analysed results of instrumentation. It is reasonable to assume that the ground control measures as proposed will require modification and refinement as experience is gained in this mining operation.

The recommended ground support is as follows:

### 7.2 Hangingwall

The hangingwall support consists of 16 mm diameter, untensioned, full-column grouted cable bolts installed in down-holes from the hangingwall drift to provide a 1.8 x 1.8 metre pattern at the mining

limit. The typical layout is shown in Figures 45 and 46. The purpose of this support is to preserve and enhance rock mass cohesion within the hangingwall and limit the development of induced fracture zones.

**7.3 Overcuts and Undercuts**

i) 3 metre long grouted rebars in the backs on a 1.2 x 1.2 m pattern.

ii) 2 metre long grouted rebars in the walls on a 1.2 x 1.2 m pattern.

iii) Standard wire mesh screen on the backs.

**7.4 Access Drifts**

2 metre long grouted rebars in the walls and backs on a 1.2 x 1.2 m pattern.

**7.5 Drawpoint Crosscuts**

3-metre long grouted rebars on a 1.2 x 1.2 m pattern up to 10 metres back from the brow area and 2-metre long grouted rebars on a 1.2 x 1.2 m pattern on the walls and remaining back area. The 3-metre long rebars should be inclined at an uphole angle of approximately 55°N to intersect the critical release surface.

**7.6 Procedure for Locally Poor Ground**

Apply a nominal layer of shotcrete where necessary in areas of broken or locally poor ground to prevent unravelling and spalling of loose.

## 8. INSTRUMENTATION

The purpose of monitoring a new mining method is:

a) to provide a warning system for ground instability.

b) to obtain data so predicted ground conditions from numerical modelling can be modified to represent actual ground behaviour more closely.

The following types of instruments have been installed in the Onaping No. 3 orezone:

### 8.1 Borehole extensometers

The Geokon Model A-3 extensometer is made of four basic components.

1) a groutable rebar anchor
2) 8 mm diameter stainless steel measurement rods
3) 8 mm schedule 40 PVC protective tubing
4) extensometer head assembly.

The arrangement of these components is shown in Figure 47.

The extensometer can be used in vertical or inclined boreholes which can be be filled with grout. The borehole anchors are made from lengths of rebar connected to the measurement rods. The rods are protected from the grout by plastic pipe which may be lubricated internally to ensure free travel.

The rods follow the anchor movements which are sensed by measuring the position of the upper rod tip with reference to a plate

the instrument head. Up to six of these combinations may be installed in one NX size borehole for measuring the magnitude of any movements and the location of any movement zones.

Assembly and borehole installation is completed as per Figure 47. With a polyethelene grout pipe attached to the lowest anchor, grouting is accomplished using neat Portland No. 2 cement mixed with water in a 1:1 ratio. As grouting proceeds the grouting pipe is slowly withdrawn from the borehole.

Figure 48 shows a typical arrangement of the extensometer head when linear potentiometers are used to provide remote readout. Readings from the potentiometers are transmitted via multi-conductor shielded signal cable to a remote station. Readout is accomplished by using a digital multimeter showing the electrical resistance of the potentiometer and rod movement calculated from a calibration curve.

## 8.2 Ground Movement Monitors

The Ground Movement Monitor (G.M.M) is an electrical extensometer capable of measuring the surface movements of an underground opening relative to the anchor point of the rockbolt to which it is attached. (Figure 49)

The casing is constructed from aluminum hexagonal stock, which houses a linear potentiometer as the measuring element. The casing is screwed onto the exposed end of a conventional mechanical rockbolt.

The centre shaft of the potentiometer is fixed to the mouth of the borehole. Any deformation at the rockface is converted into an electrical resistance change which is measurable by a digital multimeter.

## 8.3 Cablebolt load cells

Cablebolt load cells are cylindrical shells with four strain gauges aligned perpendicularly around the periphery of the shell. The body is enclosed within an outer case, the unit can be sandwiched between two bearing plates and a cablebolt passed through its centrehole (Figure 51). Installations are made at the collar locations of toe-grouted cablebolts aligned perpendicularly to the hangingwall on every third ring of cables. Toe-grouting occurs to an approximate height of 4 metres and the cable is nominally tensioned and clamped above the upper bearing plate. Load applied to the bearing plates is sensed as strain within the cell body, which indicates the magnitude of the applied load.

The readout monitors the change in electrical resistance of the strain gauges and is measured using a digital multimeter. The installed arrangement shown in Figure 50 is achieved by using a hydraulic centre-pull jack, pulling stand and two cablebolt clamps.

## 8.4 Blast Vibration Monitoring

A program would be initiated to measure ground vibrations from production blasting. Suitable seismographical-type instrumentation such as the The Falconbridge Blast Vibration Monitoring System or any similar commercially available system used.

The objectives of the monitoring system are as follows:

i) Establish engineering guidelines for reliable prediction of vibration levels in advance of blasting operations.

ii) Establish permissible levels of ground vibration by co-relating extent of damage due to blasting with vibrations.

iii) Provide operators with information on the effectiveness of the blast design, i.e. initiation sequence, occurence of missfires or cut-offs and other detonation irregularities.

iv) Investigate the feasibility of relating the rock mass dynamic constants to deteriorating ground conditions during the mining cycle.

The Falconbridge blast vibration monitoring system has been developed over a number of years. The present system consists of a four channel cassette recording unit, triaxial geophones and a separate playback - printout unit. The system is capable of monitoring both seismic velocity and acceleration. The available recording time is 25 minutes of tape play, with an optional delay of up to 20 minutes prior to recording.

## 8.5 Instrumentation concept and layout

The plan locations of the instrumentation are shown in Figures 45 and 46. These are:

i) Three 4-point anchor rod extensometers installed through the hangingwall from the hangingwall drift and perpendicular to the

mining limit. Anchors are located at 3, 6, 7.6 and 9.1 metres from the collar. The objective of the extensometer stations is to locate and monitor potential hangingwall displacement during the mining cycle.

ii) Three observation boreholes located alongside each extensometer installation. The objective is to monitor the hangingwall periodically using a borehole camera and correlate findings to the extensometer results.

iii) Ten cablebolt load cells, located at the collar of the cablebolts which are mutually perpendicular to the mining limit on the cablebolt rings as shown. Each cable is toe grouted for 3 metres to be nominally tensioned during installation of the load cells. The objective is to monitor change in load and establish load behaviour and ground reaction curves for the cablebolt support system in response to mining activity.

iv) Thirteen ground movement monitors (G.M.M.'s) installed on 3.6 metre long rockbolts above the backs of the first panel undercut and overcut. Eight G.M.M.'s are located in the overcut, and five in the undercut in a staggered pattern on either side of the stope centreline. The objective is to monitor back displacement in response to the formation of a stress arch as mining progresses.

v) It is intended to monitor each production blast as part of the blasting evaluation program. Geophones located in the hangingwall drift, at a constant separation and distance from the production blast, will also monitor any progressive change in the hangingwall rock mass dynamic elastic constants during mining.

## 9. DRILLING AND BLASTING

### 9.1 Introduction

This portion of the study details drilling and blasting design for the orebody. Included in the plans are details of hole diameter, drilling pattern, hole charging and initiation sequence.

The requirement of a suitable degree of fragmentation for material handling systems is emphasized. The importance of maintaining the integrity of the incompetent hangingwall is also a major design concern.

Particular consideration is given to incorporating controlled rock throw or directional heaving techniques, perimeter blasting methods and ground vibration control requirements.

The design is based on developments in the field of blasting technology, empirical data analysis and experience from operating mines.

### 9.2 Design Criteria

The mining method, the geometry of the orebody and the condition of the wall rocks, require special design considerations.

Design criteria have been established as follows:

9.2.1 Fragmentation

A suitable degree of fragmentation will require ninety percent (90%) of the ore broken to minus 30 cm size. This size should be satisfactory for LHD mucking, grizzly by-pass arrangement and coveyor belt handling.

Controls must be exercised to ensure minimal oversize material requiring secondary blasting and handling.

These criteria will require detailed consideration of factors that influence the rock breakage mechanism.

9.2.2 Controlled Rock-throw

The mining method will displace broken ore from the footwall area by the blasting process to the centre of the stope where it will run to the mucking elevation. Directed rockthrow is necessary due to the majority of the footwall being flatter than 55°, which is considered to be the minimum angle for free rockflow.

Ore-loss on the footwall can be minimized by:

a) sequencing the initiation to direct the throw of rock away from the footwall

b) blasting an artificial footwall to $\geq$ 55°

c) a combination of a) and b)

9.2.3 Control of Overbreak

Overbreak will be a critical factor particularly to the hangingwall and the brows of the undercut drawpoints.

Attention must be directed towards:

a) Concern in exposing remote control mucking equipment to potential damage by the fall of loose.

b) The stability of the overcut brow - to avoid overbreak and collar loss of pre-drilled holes.

c) The stability of the bench face - which must be vertical to avoid overbreak and subsequent loading difficulties.

d) The stability of the undercut back and brow - to maintain efficient drawpoint access.

e) The stability of the hangingwall drift - to maintain access for fill placement and instrument monitoring.

With reference to these points the following topics were noted as areas of concern by operating personnel:

i) structural examination of the rock mass indicates that the hangingwall contact is bounded by a lower RQD zone, (50-60% - poor to fair competancy), which is approximately 2 metres wide.

ii) fracture frequency spacing can occur up to 10 per metre locally.

iii) computer simulation predicts tensile shear zones extending up to 12 metres into the hangingwall.

iv) development mining has shown that the hangingwall norites are "very blocky", with an average block size of 1 metre.

9.2.4 Ground vibrational control

It is considered necessary to minimise the effect of blasting on mine structures, particularly the following areas:

1. the hangingwall, including the support systems.
2. the stope undercut area.
3. the drawpoints.
4. other critical openings such as the footwall drifts and access cross-cuts.

These criteria require that the design take into account the intensity of ground vibrations due to blasting.

9.2.5 Design Alternatives for Borehole Layout

(i) fanned holes drilled vertically from the footwall and inclined towards the hangingwall. Blind holes on the footwall will be blasted to the ore outline (approx. 40°). Blasting will be in two stages:

a) blasting the hangingwall holes to direct rockthrow into the hangingwall void

b) mucking out from the drawpoint and creating a void for the second- stage footwall blasting.

(ii) An artificial footwall drilled at approximately 55° to ensure that the blasted ore will run into the

undercut. This may require additional holes drilled from the overcut for recovery of the foot- wall wedge of ore. Problems could occur with this alternative with the blasting, separation and dilution of waste at the undercut-footwall elevation.

Alternatives include:

a) blasting and mucking waste separately.

b) blasting waste and leaving it in the void.

Problem areas requiring consideration are:

a) the stability of the overhang created by the blasting of the waste wedge.

b) possible loss of ore along the footwall due to mis-directed throw.

c) problems associated with charging and loading a complex blasting sequence.

## 9.3 General Description of the Design

Drilling and blasting of the orebody will be carried out from an overcut or upper sublevel.

Production drilling plans call for the use of an electric hydraulic blasthole drill. Drill hole size will be 113 mm in diameter and will range in length from 5 m to 18 m.

A 3-metre square raise will be established in the east end of the first panel.

A benching method will be used to break out the slot and to retreat the production holes away from the slot. Two or three rows will be a normal blast. The centre holes will be fired first, followed by the hangingwall holes and the footwall holes last. The initiation sequence is designed to direct ore from the hangingwall along the strike of the stope, and from the footwall area, normal to the strike.

It is expected that the ore in the hangingwall area will be sufficiently displaced to provide a void for the footwall muck. However it is possible that some mucking from the hangingwall area may be necessary prior to blasting the footwall holes.

9.3.1 Design Factors

The following factors were considered in the design:

1. Rock physical properties and Structural geology: The properties considered were the compressive and tensile strengths and features such as the fracture and joint systems of the orezone.
2. The slot raise was driven by conventional timber-raising technique due to its relatively short length and required three days to complete. Drop-raising with a sequential crater-type cut utilising 114 mm holes was an alternative considered.
3. The production target is limited to 360 tonnes per day

due to tramming capacity on the 3875 level. Over two shifts, 180 tonnes per shift, will require 60 - 80 cars trammed per shift depending upon the achieved fragmentation. 1 - 2 lines will have to be blasted per week to produce the 900 tonnes of broken ore required.

4. Blasthole pattern:

   (a) Drill hole spacing, burden dimensions and ratios as well as the pattern were included in this study. Hole pattern options considered include, square, rectangular and staggered designs.

   (b) The drilling pattern should be simple and avoid drill offsets in two dimensions during set up.

   (c) The inclination of the blastholes was considered and two alternatives appear to be feasible:

      i) All holes drilled parallel or parallel to the dip of the orebody as shown in Figure 52

      ii) Holes in the footwall area drilled vertical or near vertical while those near the hangingwall drilled in a fan-like pattern as shown in Figure 53.

5. Loading Detail:

a) the design will utilize in-hole primed, decked and delayed charges.

b) a 7.6 cm/sec maximum peak particle velocity at a distance of 30 metres will minimise vibration damage.

c) a charge weight per delay of approximately 38 kg of ANFO is required for production holes.

d) a charge weight per delay of approximately 20 kg of ANFO is required for the hangingwall perimeter holes.

e) the use of decoupled perimeter charge or low strength polystyrene/ANFO will be investigated as a measure to reduce overbreak.

6. Initiation of priming systems.

7. Initiation sequence.

## 9.4 Design Detail

The blasting design details for Onaping No. 3 Zone are discussed in the following areas:

i) the slot blast.

ii) production blasting using parallel holes.

iii) production blasting using fanned holes.

Parallel holes and fanned holes are regarded as viable alternatives. The intention is to assess the success of both alternatives

when mining the first panel of the orebody. Upon completion of the slot, the first half of the stope will be mined using parallel blastholes, and a fanned blasthole pattern in the second half. The vibration level, fragmentation and rockthrow of each pattern will be monitored and assessed in selecting the design for subsequent mining.

9.4.1 The Slot

For reasons outlined below, the raise-to-slot opening is designed to be removed in three stages. These consist of two vertical bench lifts and final sill removal.

i) Safety - due to the risks involved in vertical face benching where workers must load holes in close proximity to large vertical openings.

ii) Ease of design - the use of a three-stage slot blast in the design. This reduces the risk of loading errors and the chance of misfires.

Slot Stage I

| | |
|---|---|
| Hole Dia | - 114 mm |
| No. Holes Fired | - 12 |
| No. Delays Used | - 18 (26 Decks) |
| Type of Explosive | - ANFO - Normal Holes<br>POLYSTYRENE ANFO-H.W.control holes |
| Type of Delay | - All delays in the hole. Nonel short period. Nos.0,2,3,5,7,9,11,13,15,18, 21,23,25 and long period Nos. 9,11, 13,26,18. |
| Method of Initiation | - Non-electric using E cord, or similar, downlines and sliders. |

| | |
|---|---|
| Method of Loading | - Pneumatic |
| Stemming | - 1st Choice - Drill Cuttings<br>- 2nd Choice - Sand |
| Plug | - Wooden, two piece |
| Supporting Rope | - Carry Strap |
| Explosive Load | - ANFO 10.3 kg/m<br>Polystyrene ANFO 5.71 kg/m |
| Max. Charge Wt. | - Normal Holes 40.8 kg/delay<br>Control Holes 20.4 kg/delay |
| Volume Broken | - 635 $m^3$ |
| Collar Stemming Length | - 2.4 metres |
| Total Pounds Explosive | - 472 kg ANFO equivalent |
| Powder Factor | - 1.35 $kg/m^3$ |
| Holes Fired | - 31E--K,32K,33I--K,34I,35F--H |

9.4.2 Production blasting using parallel holes

The pattern for production blasts can be considered in two distinct stages. Stage 1, the ore adjacent to the hangingwall and the control holes which delineate the hangingwall limits are blasted first to a depth of two lines. Stage 2, the footwall ore is then blasted off the footwall contact.

The number of holes fired in stage 2 may be varied to suit production requirements.

Production blast - Stage 1

| | |
|---|---|
| Hole diameter | - 114 mm |
| No. holes fired | - 10 |

| | |
|---|---|
| No. delays | - 13 (37 decks) |
| Type of explosive | - ANFO - production holes<br>POLYSTYRENE ANFO - control holes |
| Type of delay | - NONEL - short period 1,2,3,5,7,9,11,13 15,18,21, 24<br>- long period 10 |
| Method of initiation | - Non-electric using E-cord, or similar, downlines and slider primers. |
| Method of loading | - Pneumatic |
| Stemming | - Drill cuttings |
| Plug | - Wooden, two piece |
| Supporting rope | - Carry strap |
| Explosive load | - ANFO 10.25 kg/m<br>POLYSTYRENE ANFO 5.64 kg/m |
| Max. charge weight | - Production holes 40.3 kg/delay<br>Control holes 20.4 kg/delay |
| Volume broken | - 441 $m^3$ |
| Total weight of explosive | - 1011 kg ANFO equivalent |
| Powder Factor | - 2.29 $kg/m^3$ |
| Holes fired | - 29N-S, 30S, 28S, 27N-S |

Holes 30S and 29S should have a surface trunk line attached to the collar delays. This avoids cap scatter affecting the firing sequence and the risk of hole 29S firing before 30S. The same situation exists for holes 27P, 27N, 27R, 27S and 28S.

Production blast - Stage 2

To blast the ore from the footwall.

| | |
|---|---|
| Hole diameter | - 114 mm |
| No. holes fired | 5 |
| No. Delays used | 12 (21 decks) |

| | |
|---|---|
| Type of Explosive | ANFO Normal holes<br>AQUAMEX Blind wet holes |
| Type of Delay | NONEL - Short period No. 1,2,3,5,7,9<br>11,13,15,18,21,23 |
| Explosives Load | ANFO 10.25 kg/m<br>AQUAMEX 16 kg/m |
| Max. charge Weight | 40.8 kg/delay |
| Volume broken | 204 $m^3$ |
| Total weight expl. | 637 kg ANFO equivalent |
| Powder Factor | 3.12 $kg/m^3$ |
| Holes Fired | 27A--M, 29A--L |

9.4.3 Production blasting using fanned holes

Blasting remains a two stage operation with the same comments as for parallel holes.

Production blast - Stage 1

| | |
|---|---|
| Hole diameter | 114 mm |
| No. Holes Fired | 9 |
| No. delays used | 16 (25 decks) |
| Type of Explosive | ANFO Normal Holes<br>POLY ANFO control holes |
| Type of Delay | NONEL Short period Nos. 1,2,3,5,7,9,11,<br>13,15,18,21,23,25<br>Long period Nos. 9,11, 13 |
| Explosive Load | ANFO 1025 kg/m<br>POLYANFO 5.64 kg/m |
| Volume broken | 382 $m^3$ |
| Total explosive weight | 773 kg |

| | |
|---|---|
| Powder Factor | 2.4 kg/$m^3$ |
| Holes Fired | 26G--F, 25I, 24 G--F |

Production Blast - Stage 2

| | |
|---|---|
| Hole diameter | 114 mm |
| No. Holes fired | 10 |
| Delays used | 18 (18 Decks) |
| Type of explosive | AQUAMEX - blind holes |
| Types of delay | NONEL short period nos. 1,2,3,5,7,9,11 13,15,18,21<br>Long period Nos. 5,7,9,11,13,16,18 |
| Explosive load | AQUAMEX 16 kg/m |
| Volume ground broken | 280$m^3$ |
| Total explosive weight | 5.12 kg ANFo equivalent |
| Powder Factor | 1.8 kg/$m^3$ |
| Holes fired | 26A--E, 24A--E |

Note: That the collar delays should be buried in the collar stemming for protection. All holes with surface delays should be double primed for safety.

Where the actual length varies from the theoretical length each deck should be increased or decreased in size proportionately.

The overloading of decks 35H&U is permissible in this isolated example due to the use of polystyrene ANFO.

The following factors have been incorporated into the blast design:

i) Delay Numbers - There are five considerations

a) $\pm$ 10% cap scatter

b) All delay elements must be burning before the first charge detonates.

c) Each charge must have two free faces to break to.

d) The maximum charge weight per delay must not be exceeded.

e) At least 12 millisecond intervals between charges at the maximum weight per delay must be allowed to avoid additive effects.

The only exception to this rule is when the burden is minor.

ii) Charge Distribution - The aim is to provide an even distribution of explosive throughout the rock mass. This is achieved by using smaller explosive weight per deck, and increasing the number of decks.

In the polystyrene ANFO control holes the plug has been set lower to avoid leaving a rock protrusion in the hanging-wall.

iii) Vibration Levels - The charge weights and delays selected are extremely conservative concerning the vibration levels produced. From readings at Lockerby it is not expected that any of the blasts will exceed 7.5 cm/sec p.p.v. from a vector sum.

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# APPENDIX A

## STRUCTURAL DATA - ONAPING NO. 3 ZONE

## APPENDIX

## STRUCTURAL DATA - ONAPING No. 3 ZONE

Joints in Dark Norite Breccia

No. of data points - 85.

| Dip Direction | Dip | Dip Direction | Dip | Dip Direction | Dip |
|---|---|---|---|---|---|
| 308 | 50 | 300 | 60 | 206 | 30 |
| 42 | 45 | 217 | 87 | 306 | 70 |
| 278 | 80 | 262 | 20 | 183 | 70 |
| 261 | 20 | 327 | 85 | 278 | 80 |
| 301 | 45 | 260 | 30 | 308 | 75 |
| 336 | 75 | 317 | 65 | 314 | 65 |
| 327 | 40 | 314 | 70 | 318 | 65 |
| 289 | 80 | 280 | 30 | 244 | 50 |
| 308 | 85 | 312 | 85 | 325 | 60 |
| 272 | 20 | 113 | 80 | 275 | 40 |
| 341 | 70 | 263 | 87 | 42 | 20 |
| 301 | 75 | 202 | 70 | 320 | 65 |
| 263 | 75 | 220 | 70 | 242 | 30 |
| 282 | 40 | 204 | 35 | 251 | 35 |
| 316 | 40 | 3 | 65 | 282 | 75 |
| 291 | 70 | 299 | 25 | 297 | 70 |
| 246 | 20 | 159 | 70 | 312 | 60 |
| 312 | 75 | 280 | 80 | 275 | 80 |
| 300 | 60 | 102 | 60 | 263 | 30 |
| 336 | 75 | 298 | 45 | 315 | 65 |
| 296 | 60 | 320 | 75 | 325 | 70 |
| 285 | 80 | 292 | 30 | 312 | 75 |
| 140 | 85 | 286 | 80 | 313 | 45 |
| 326 | 40 | 312 | 85 | 126 | 40 |
| 336 | 75 | 111 | 30 | 156 | 87 |
| 243 | 30 | 328 | 60 | 322 | 65 |
| 199 | 35 | 12 | 35 | 300 | 70 |
| 280 | 40 | 230 | 40 | 302 | 40 |
| | | | | 294 | 70 |

## Joints in Late Granite Breccia

No. of data points - 39.

| Dip Direction | Dip | Dip Direction | Dip | Dip Direction | Dip |
|---|---|---|---|---|---|
| 339 | 70 | 315 | 70 | 307 | 70 |
| 289 | 85 | 317 | 80 | 305 | 75 |
| 330 | 65 | 154 | 80 | 328 | 65 |
| 340 | 40 | 318 | 70 | 310 | 70 |
| 331 | 75 | 326 | 60 | 323 | 20 |
| 358 | 60 | 302 | 30 | 367 | 70 |
| 305 | 40 | 291 | 40 | 281 | 35 |
| 327 | 70 | 321 | 70 | 313 | 70 |
| 335 | 80 | 330 | 80 | 306 | 75 |
| 338 | 40 | 160 | 80 | 325 | 65 |
| 330 | 75 | 319 | 70 | 311 | 70 |
| 326 | 60 | 327 | 60 | 322 | 87 |
| 289 | 85 | 313 | 20 | 302 | 30 |

## Joints in Feldspathic Gneiss

No. of data points - 96.

| Dip Direction | Dip | Dip Direction | Dip | Dip Direction | Dip |
|---|---|---|---|---|---|
| 327 | 65 | 270 | 85 | 295 | 87 |
| 303 | 70 | 344 | 45 | 320 | 70 |
| 310 | 40 | 346 | 70 | 262 | 80 |
| 357 | 45 | 0 | 85 | 54 | 20 |
| 320 | 35 | 146 | 30 | 282 | 40 |
| 346 | 70 | 333 | 70 | 324 | 70 |
| 325 | 85 | 49 | 20 | 7 | 80 |
| 150 | 30 | 223 | 70 | 357 | 80 |
| 353 | 70 | 324 | 70 | 306 | 45 |
| 178 | 70 | 11 | 80 | 311 | 40 |
| 320 | 25 | 9 | 75 | 357 | 45 |
| 300 | 85 | 299 | 50 | 327 | 40 |
| 256 | 20 | 304 | 80 | 346 | 70 |
| 320 | 70 | 325 | 40 | 338 | 80 |
| 2 | 70 | 271 | 85 | 356 | 45 |
| 8 | 75 | 337 | 60 | 353 | 70 |
| 309 | 45 | 345 | 70 | 351 | 65 |
| 305 | 80 | 3 | 85 | 311 | 80 |
| 316 | 40 | 302 | 85 | 152 | 80 |

Joints in Feldspathic Gneiss

No. of data points - 96.

| Dip Direction | Dip | Dip Direction | Dip | Dip Direction | Dip |
|---|---|---|---|---|---|
| 316 | 20 | 358 | 75 | 349 | 60 |
| 292 | 35 | 352 | 65 | 344 | 70 |
| 343 | 40 | 317 | 30 | 345 | 70 |
| 13 | 75 | 270 | 45 | 124 | 20 |
| 339 | 70 | 314 | 20 | 302 | 85 |
| 313 | 80 | 279 | 65 | 350 | 75 |
| 285 | 65 | 223 | 45 | 318 | 30 |
| 333 | 60 | 324 | 50 | 50 | 30 |
| 310 | 40 | 161 | 60 | 315 | 20 |
| 346 | 70 | 315 | 70 | 211 | 15 |
| 338 | 80 | 287 | 65 | 220 | 45 |
| 280 | 70 | 324 | 60 | 4 | 75 |

Faults in No. 3 Zone

No. of data points - 8.

| Dip Direction | Dip |
|---|---|
| 339 | 80 |
| 311 | 50 |
| 268 | 65 |
| 308 | 65 |
| 338 | 70 |
| 271 | 65 |
| 325 | 70 |
| 279 | 80 |

# APPENDIX B

# LABORATORY TEST RESULTS

TABLE 8

| Sample | Rocktype | Sample Length (mm) | Sample Diameter (mm) | L/D | Remarks |
|---|---|---|---|---|---|
| 1-F-1A | Felsic Gneiss | 76.8 | 32.4 | 2.37 | Explosive failure. Multiple failure planes parallel to longitudinal axis. |
| 2-F-1A | Felsic Gneiss | 77.2 | 32.5 | 2.38 | Explosive failure. Multiple failure planes parallel to longitudinal axis. |
| 3-F-1A | Felsic Gneiss | 75.2 | 32.4 | 2.32 | Failed in two planes inclined at 20° to longitudinal axis (probably pre-existing) |
| 4-F-1A | Felsic Gneiss | 76.4 | 32.4 | 2.36 | Split along multiple longitudinal planes. Explosive failure. |
| 5-F-1A | Felsic Gneiss | 76.4 | 32.4 | 2.36 | Split along multiple longitudinal planes. Explosive failure. |
| 1-L-1A | Mafic Gneiss | 76.8 | 32.4 | 2.37 | Split along multiple longitudinal planes. Explosive failure. |
| 2-L-1A | Mafic Gneiss | 77.2 | 32.5 | 2.38 | Split along multiple longitudinal planes. Explosive failure. |
| 3-L-1A | Mafic Gneiss | 75.5 | 32.4 | 2.33 | Split along multiple longitudinal planes. Explosive failure. |
| 4-L-1A | Mafic Gneiss | 77.7 | 32.5 | 2.39 | Split along multiple longitudinal planes. Explosive failure. |
| 5-L-1A | Mafic Gneiss | 76.6 | 32.1 | 2.39 | Failed along single plane inclined at 20° to longitudinal axis (probably pre-existing). |

TABLE 8 (cont'd)

| Sample | Rocktype | Sample Length (mm) | Sample Diameter (mm) | L/D | Remarks |
|---|---|---|---|---|---|
| 1-DB-1A | Dark Norite | 75.6 | 32.4 | 2.33 | Stable failure along 12° plane |
| 2-DB-1A | Dark Norite (with pyrite & chalcopyrite) | 76.2 | 32.5 | 2.34 | Stable failure (some slabbing) |
| 2-DBX-1A | Mineralised Dark Norite Breccia | 75.0 | 32.4 | 2.38 | Stable failure (some slabbing) |
| 3-DB-1A-1 | Mineralised Dark Norite | 77.0 | 32.4 | 2.38 | Stable failure (some slabbing) |
| 3-DB-1A | Mineralised Dark Norite | 76.4 | 32.8 | 2.33 | Stable failure along longitudinal planes. |
| 4-DB-1A-1 | Mineralised Dark Norite | 75.1 | 32.5 | 2.31 | Stable failure (some slabbing) |
| 1-DN-1A | Dark Norite | 76.5 | 32.4 | 2.36 | Stable failure (some slabbing). |
| 2-DN-1A | Dark Norite | 79.2 | 32.5 | 2.44 | Stable failure along 10° planes. |
| 2-DNX-1A | Dark Norite | 75.4 | 32.5 | 2.32 | Stable failure along longitudinal planes. |
| 3-DN-1A | Dark Norite | 77.3 | 32.5 | 2.38 | Stable failure along longitudinal planes. |
| 4-DN-1A | Dark Norite | 77.9 | 32.5 | 2.40 | Stable failure along longitudinal planes. |

TABLE 9

| Sample | Compressive Strength $C_\tau$ (MPa) | Secant Modulus (at 50% of each load) $E_{\emptyset, 5\tau}$ ($10^4$ MPa) | Tangent Modulus (at 50% of peak load) $E_{t,50}$ ($10^4$ MPa) | Modulus of Resilience R (MPa) | Poisson's Ratio | Volumetric strain (at 50% of peak load) $\frac{DV}{V}_{,50}$ ($\times 10^{-3}$) |
|---|---|---|---|---|---|---|
| 1-F-1A | 104 | 2.29 | 2.82 | .193 | .06 | 2.01 |
| 2-F-1A | 184 | 2.59 | 4.13 | .410 | .06 | 3.17 |
| 3-F-1A | 76.4 | 2.05 | 2.51 | .116 | .04 | 1.71 |
| 4-F-1A | 236 | 2.52 | 4.01 | .694 | .06 | 4.09 |
| 5-F-1A | 136 | 2.16 | 3.52 | .263 | .05 | 2.85 |
| 1-L-1A | 114 | 1.91 | 3.15 | .206 | .04 | 2.74 |
| 2-L-1A | 357 | 3.72 | 5.43 | 1.17 | .08 | 4.01 |
| 3-L-1A | 129 | 2.52 | 3.41 | .242 | .04 | 2.33 |
| 4-L-1A | 151 | 1.86 | 3.48 | .328 | .11 | 3.12 |
| 5-L-1A | 97 | 1.90 | 3.11 | .151 | .05 | 2.30 |
| 1-DB-1A | 93.4 | 2.28 | 3.06 | .143 | .06 | 1.80 |
| 2-DB-1A | 77.0 | 2.09 | 2.75 | .108 | .09 | 1.49 |
| 2-DBX-1A | 50.4 | 1.25 | 1.75 | .072 | .04 | 1.87 |
| 3-DB-1A | 88.6 | 2.43 | 3.18 | .123 | .07 | 1.56 |
| 4-DB-1A-1 | 82.0 | 2.62 | 3.39 | .099 | .06 | 1.37 |
| 4-DB-1A-2 | 77.1 | 2.14 | 2.94 | .101 | .05 | 1.60 |
| 1-DN-1A | 129 | 1.64 | 3.83 | .217 | .03 | 3.68 |
| 2-DN-1A | 119 | 2.31 | 3.16 | .224 | .08 | 2.20 |
| 2-DNX-1A | 129 | 2.62 | 3.83 | .217 | .06 | 2.16 |
| 3-DN-1A | 145 | 3.19 | 3.82 | .274 | .10 | 1.82 |
| 4-DN-1A | 118 | 1.68 | 3.23 | .216 | .05 | 3.27 |

* See Table 8 for rock types

TABLE 10 - TRIAXIAL COMPRESSION TESTS

| SAMPLE TYPE | FAILURE ANGLE Θ (°) | PREDICTED INTERNAL FRICTION ANGLE Ø (°) | σ 1 (MPa) | σ 3 (MPa) | COHESION C, (MPa) | Ø (°) | REMARKS |
|---|---|---|---|---|---|---|---|
| DARK NORITE | 58 | 26 | 231 | 6.9 | | | |
| | 66 | 42 | 253 | 13.8 | | | |
| | 62 | 34 | 261 | 20.7 | | | |
| | Avg-62 | Avg-34 | | | | | |
| | Mohr-Coulomb plotted values | | | | 57.9 | 32 | |
| MINERALISED DARK NORITE | 69 | 48 | 131 | 6.9 | | | |
| | 68 | 46 | 175 | 13.8 | | | |
| | 73 | 56 | 224 | 20.7 | | | |
| | Avg-70 | Avg-50 | | | | | |
| | Mohr-Coulomb plotted values | | | | 15.9 | 48 | |
| LATE GRANITE BRECCIA | Apparent Vertical Fractures (90°) | - | 176 | 6.9 | | | No Shear |
| | | - | 218 | 13.8 | | | |
| | | - | 273 | 20.7 | | | Visible |
| | Mohr-Coulomb plotted values | | | | 24.1 | 49 | |
| DARK NORITE BRECCIA | 69° | 48 | 178 | 6.9 | | | Joint |
| | 76° | 62 | 241 | 13.8 | | | Joint |
| | 60° | 30 | 282 | 20.7 | | | |
| | Avg-68 | Avg-47 | | | | | |
| | Mohr-Coulomb plotted values | | | | 23.4 | 51 | |
| FELDSPATHIC GNEISS | Vertical (90°) | - | 74 | 6.9 | | | |
| | 61 | 32 | 212 | 13.8 | | | |
| | Vertical (90°) | - | 138 | 20.7 | | | |
| | Mohr-Coulomb plotted values | | | | Insufficient sample material/data available. | | |

TABLE 11 TENSION TESTS

| SAMPLE TYPE | FAILURE LOAD (kg) | CONTACT AREA ($cm^2$) | FAILURE STRESS (psi) / (MPa) | REMARKS |
|---|---|---|---|---|
| DARK NORITE | 1247 | 14.0 | 1268/8.74 | Near horizontal failure plane; no jointing visible |
| MINERALISED DARK NORITE | 1044 | 14.2 | 1046/7.21 | Near horizontal failure plane; no jointing visible |
| LATE GRANITE BRECCIA | 1361 | 14.1 | 1370/9.45 | Near horizontal failure plane; no jointing visible |
| DARK NORITE | 953 | 16.6 | 818/5.64 | Intersecting joint fracture angled at 39° and 27° to horizontal |
| FELDSPATHIC GNEISS | 1418 | 14.5 | 1396/(9.63) | Near horizontal failure plane; no jointing visible |

TABLE 12 ELASTIC MODULI DETERMINATIONS

| SAMPLE TYPE | POISSON'S RATIO | YOUNG'S MODULUS (E), ($MPa \times 10^4$) |
|---|---|---|
| DARK NORITE | 0.30 | 8.55 |
| MINERALISED DARK NORITE | 0.25 | 2.90 |
| LATE GRANITE BRECCIA | 0.26 | 4.96 |
| DARK NORITE BRECCIA | 0.23 (5) | 5.31 |
| FELDSPATHIC GNEISS | 0.28 | 5.86 |

# APPENDIX C

# ROCK QUALITY DETERMINATION

## APPENDIX C

## ROCK QUALITY DETERMINATION - ONAPING NO. 3 ZONE

Borehole No. 37-095
Core diameter AXT

| Footage | RQD (estimated) | RQD (actual) |
|---|---|---|
| 0 - 25 | 70% | 89% |
| 25 - 50 | 70 | 89 |
| 50 - 75 | 70 | 82 |
| 75 -100 | 60 | 63 |
| 100 -103 | 60 | 70 |

Borehole No. 37-093
Core Diameter AXT

| Footage | RQD (estimated) | RQD (actual) |
|---|---|---|
| 0 - 25 | 65% | 75% |
| 25 - 54 | 50 | 62 |
| 54 - 79 | 55 | 71 |
| 79 -104 | 75 | 87 |
| 104 -115 | - | 80 |

Borehole No. 35-094
Core diameter AXT

| Footage | RQD (estimated) | RQD (actual) |
|---|---|---|
| 0 - 25 | 75% | 85% |
| 25 - 50 | 95 | 89 |
| 50 - 75 | 55 | 87 |
| 75 -100 | 40 | 74 |
| 100 -125 | 50 | 79 |
| 125 -150 | 40 | 83 |
| 150 -175 | 60 | 77 |
| 175 -200 | 55 | 77 |
| 200 -219 | 55 | 68 |

Borehole No. 35-095
Core diameter AXT

| Footage | RQD (estimated) | RQD (actual) |
|---|---|---|
| 0 - 25 | 75% | 84% |
| 25 - 50 | 70 | 83 |
| 50 - 75 | 80 | 82 |
| 75 -100 | 65 | 65 |
| 100 -125 | 80 | 73 |
| 125 -149 | 65 | 66 |
| 149 -175 | 85 | 87 |
| 175 -200 | 75 | 69 |
| 200 -214 | 90 | 97 |

Borehole No. 37-1
Core Diameter BX

| Footage | RQD (estimated) | RQD (actual) |
|---|---|---|
| 0 - 19 | 30% | 72% |
| 19 - 38 | 40 | 68 |
| 38 - 48 | 50 | 80 |

Borehole No. 37-2
Core Diameter BX

| Footage | RQD (estimated) | RQD (actual) |
|---|---|---|
| 0 - 18 | 50% | 46% |
| 18 - 37 | 65 | 78 |
| 37 - 53 | 60 | 73 |

Borehole No. 37-4
Core Diameter BX

| Footage | RQD (estimated) | RQD (actual) |
|---|---|---|
| 0 - 19 | 55% | 64% |
| 19 - 38 | 50 | 57 |
| 38 - 56 | 50 | 48 |
| 56 - 74 | 50 | - |
| 74 - 92 | 20 | - |
| 92 - 95 | 0 | - |

# ONAPING #3

## Borehole No. 35-094

| Footage | RQD |
|---|---|
| 0 - 5' | 75% |
| 5'- 13' | 94 |
| 13'- 24' | 82 |
| 24'- 39' | 87 |
| 39'- 45' | 91 |
| 45'- 50' | 93 |
| 50'- 56' | 83 |
| 56'- 61' | 80 |
| 61'- 67' | 89 |
| 67'- 75' | 93 |
| 75'- 84.5' | 82 |
| 84.5'- 89' | 82 |
| 89'-100' | 63 |
| 100'-106' | 91 |
| 106'-109' | 52 |
| 109'-118' | 73 |
| 118'-125' | 88 |
| 125'-133' | 85 |
| 133'-139' | 85 |
| 139'-143' | 79 |
| 143'-150 | 80 |
| 150'-154' | 91 |
| 154'-161' | 82 |
| 161'-164' | 22 |
| 164'-174' | 85 |
| 174'-183' | 74 |
| 183'-190' | 63 |
| 190'-200' | 88 |
| 200'-210' | 94 |
| 210'-219' | 40 |

## Borehole No. 35-095

| Footage | RQD |
|---|---|
| 0 - 8.5' | 83% |
| 8.5'- 16.5' | 90 |
| 16.5'- 22' | 85 |
| 22'- 29.5' | 73 |
| 29.5'- 37' | 80 |
| 37'- 42' | 91 |
| 42'- 50' | 83 |
| 50'- 59' | 84 |
| 59'- 65' | 89 |
| 65'- 75' | 76 |
| 75'- 83' | 32 |
| 83'- 89.5' | 71 |
| 89.5'-94.5' | 74 |
| 94.5'-100' | 96 |
| 100'-105' | 72 |
| 105'-111' | 67 |
| 111'-115.5' | 61 |
| 115.5'-125' | 83 |
| 125'-130' | 53 |
| 130'-150' | 70 |
| 150'-154' | 88 |
| 154'-169' | 86 |
| 169'-175' | 91 |
| 175'-180' | 80 |
| 180'-188' | 81 |
| 188'-192' | 44 |
| 192'-200' | 63 |
| 200'-214' | 97 |

## ONAPING #3

### Borehole No. 37-093

| Footage | RQD |
|---|---|
| 1'-8' | 63% |
| 8'-18.5' | 81 |
| 18.5'-25' | 79 |
| 25'-34' | 77 |
| 34'-40' | 81 |
| 40'-45' | Ground Core |
| 45'-50' | 44 |
| 50'-54' | 57 |
| 54'-65 | 69 |
| 65'-79' | 72 |
| 79'-93.5' | 86 |
| 93.5'-104' | 89 |
| 104'-110.5' | 89 |
| 110.5'-115' | 67 |

### Borehole No. 37-094

| Footage | RQD |
|---|---|
| 2'-13.5' | 87 |
| 13.5'-18' | 77 |
| 18'-25' | 68 |
| 25'-29' | 74 |
| 29'-33' | 39 |
| 33'-36.5' | 72 |
| 36.5'-50' | 91 |
| 50'-62.5' | 66' |
| 62.5'-75' | 83 |
| 75'-88' | 67 |
| 88'-94' | 74 |
| 94'-100' | 72 |
| 100'-104' | 92 |
| 104'-113.5' | 96 |
| 113.5-121.1' | 90 |
| 121.1'-125' | 84 |
| 125'-131' | 88 |

### Borehole No. 37-095

| Footage | RQD |
|---|---|
| 1'-11' | 93 |
| 11'-23' | 84 |
| 23'-36' | 97 |
| 36'-47' | 84 |
| 47'-57' | 74 |
| 57'-67' | 91 |
| 67'-75' | 78 |
| 75'-80.5' | 60 |
| 80.5'-89' | 60 |
| 89'-92' | 61 |
| 92'-103' | 70 |

SIZE BX 37-1

| Length | RQD |
|---|---|
| 2'-4' | 51% |
| 4-8-1/2' | 38 |
| 8-1/2'-11' | 98 |
| 11'-15' | 97 |
| 15-19-1/2' | 77 |
| 19-1/2'-24-1/2'' | 85 |
| 24-1/2'-29-1/2' | 78 |
| 29-1/2'-34-1/2' | 48 |
| 34-1/2'-38-1/2' | 58 |
| 38-1/2'-43-1/2' | 99 |
| 43-1/2'-49' | 62 |

SIZE BX 37-2

| Length | RQD |
|---|---|
| 0-4-1/2' | 30 |
| 4-1/2'-8-1/2' | 55 |
| 8-1/2'-13-1/2' | 40 |
| 13-1/2'-18' | 62 |
| 18'-23' | 88 |
| 23'-27-1/2' | 93 |
| 27-1/2'-31' | 84 |
| 31'-37' | 54 |
| 37'-43' | 89 |
| 43'-53' | 64 |

SIZE BX 37-4

| | |
|---|---|
| 0-4' | 93 |
| 4'-9' | 74 |
| 9'-14' | 60 |
| 14'-17' | 56 |
| 17'-19' | Missing Core |
| 19'-23.5' | 72 |
| 23.5'-28' | 56 |
| 28'-33' | 68 |
| 33'-38' | 35 |
| 38'-44' | 27 |
| 44'-49' | 48 |
| 49'-56' | 66 |

# APPENDIX D

# ROCK MASS CLASSIFICATION

# C.S.I.R.O. SYSTEM

# CSIR GEOMECHANICS CLASSIFICATION OF JOINTED ROCK MASSES

## A. CLASSIFICATION PARAMETERS AND THEIR RATINGS

| PARAMETER | | | RANGES OF VALUES | | | | | | |
|---|---|---|---|---|---|---|---|---|---|
| 1 | Strength of intact rock material | Point load strength index | > 8 MPa | 4 - 8 MPa | 2 - 4 MPa | 1 - 2 MPa | For this low range - uniaxial compressive test is preferred | | |
| | | Uniaxial compressive strength | > 200 MPa | 100 - 200 MPa | 50 - 100 MPa | 25 - 50 MPa | 10-25 MPa | 3-10 MPa | 1-3 MPa |
| | Rating | | 15 | 12 | 7 | 4 | 2 | 1 | 0 |
| 2 | Drill core quality RQD | | 90% - 100% | 75% - 90% | 50% - 75% | 25% - 50% | < 25% | | |
| | Rating | | 20 | 17 | 13 | 8 | 3 | | |
| 3 | Spacing of joints | | > 3 m | 1 - 3 m | 0.3 - 1 m | 50 - 300 mm | < 50 mm | | |
| | Rating | | 30 | 25 | 20 | 10 | 5 | | |
| 4 | Condition of joints | | Very rough surfaces Not continuous No separation Hard joint wall rock | Slightly rough surfaces Separation < 1 mm Hard joint wall rock | Slightly rough surfaces Separation < 1 mm Soft joint wall rock | Slickensided surfaces or Gouge < 5 mm thick or Joints open 1 - 5 mm Continuous joints | Soft gouge > 5 mm thick or Joints open > 5 mm Continuous joints | | |
| | Rating | | 25 | 20 | 12 | 6 | 0 | | |
| 5 | Ground water | Inflow per 10 m tunnel length | None | | < 25 litres/min | 25 - 125 litres/min | > 125 litres/min | | |
| | | Ratio $\frac{\text{joint water press.}}{\text{major principal stress}}$ | OR 0 | | OR 0.0 - 0.2 | OR 0.2 - 0.5 | OR > 0.5 | | |
| | | General conditions | OR Completely dry | | OR Moist only (interstitial water) | OR Water under moderate pressure | OR Severe water problems | | |
| | Rating | | 10 | | 7 | 4 | 0 | | |

## B. RATING ADJUSTMENT FOR JOINT ORIENTATIONS

| Strike and dip orientations of joints | | Very favourable | Favourable | Fair | Unfavourable | Very unfavourable |
|---|---|---|---|---|---|---|
| Ratings | Tunnels | 0 | -2 | -5 | -10 | -12 |
| | Foundations | 0 | -2 | -7 | -15 | -25 |
| | Slopes | 0 | -5 | -25 | -50 | -60 |

## C. ROCK MASS CLASSES DETERMINED FROM TOTAL RATINGS

| Rating | 100 — 81 | 80 — 61 | 60 — 41 | 40 — 21 | < 20 |
|---|---|---|---|---|---|
| Class No | I | II | III | IV | V |
| Description | Very good rock | Good rock | Fair rock | Poor rock | Very poor rock |

## D. MEANING OF ROCK MASS CLASSES

| Class No | I | II | III | IV | V |
|---|---|---|---|---|---|
| Average stand-up time | 10 years for 5 m span | 6 months for 4 m span | 1 week for 3 m span | 5 hours for 1.5 m span | 10 min for 0.5 m span |
| Cohesion of the rock mass | > 300 kPa | 200 - 300 kPa | 150 - 200 kPa | 100 - 150 kPa | < 100 kPa |
| Friction angle of the rock mass | > 45° | 40° - 45° | 35° - 40° | 30° - 35° | < 30° |

| Borehole Footage | | RQD | Intact MPa Rock Strength | Joint Spacing | Separation of Joints | Continuity of Joints | Strike and Dip Orientation | Weathering & Groundwater | Score |
|---|---|---|---|---|---|---|---|---|---|
| 37093 1-8 | Description | 63% | | 204 | | | | | |
| | Rating | 12 | 5 | 10 | 5 | 5 | 5 | 19 | 61 |
| " 8-18.5 | Description | 81% | 104 | 178 | | | | | |
| | Rating | 14 | 5 | 10 | 5 | 5 | 5 | 19 | 63 |
| " 18.5-25 | Description | 79% | | 105 | | | | | |
| | Rating | 14 | 5 | 10 | 5 | 3 | 5 | 19 | 61 |
| " 25-34 | Description | 77% | | 116 | | | | | |
| | Rating | | | 10 | 5 | 3 | 5 | 19 | 61 |
| " 34-40 | Description | 81% | 114 | 229 | | | | | |
| | Rating | 14 | 5 | 10 | 5 | 3 | 5 | 19 | 61 |
| " 40-45 | Description | ? | | | | | | | |
| | Rating | | | | | | | | |
| " 45-50 | Description | 34% | | 152 | | | | | |
| | Rating | 7 | 5 | 10 | 4 | 3 | 5 | 19 | 53 |
| " 50-54 | Description | 57% | | 122 | | | | | |
| | Rating | 12 | 2 | 10 | 4 | 3 | 5 | 19 | 55 |
| " 54-65 | Description | 69% | | 139 | | | | | |
| | Rating | 12 | 2 | 10 | 4 | 3 | 3 | 19 | 53 |
| " 65-79 | Description | 72% | | 137 | | | | | |
| | Rating | 12 | 2 | 10 | 4 | 0 | 3 | 19 | 50 |
| " 79-93.5 | Description | 86% | 93.4 | 156 | | | | | |
| | Rating | 14 | 2 | 10 | 4 | 3 | 3 | 19 | 55 |
| " 93.5-104 | Description | 89% | | 377 | | | | | |
| | Rating | 14 | 2 | 20 | 5 | 3 | 3 | 19 | 66 |
| " 104-110.5 | Description | 88% | | 270 | | | | | |
| | Rating | 14 | 2 | 10 | 5 | 3 | 3 | 19 | 56 |
| " 110.5-115 | Description | 67% | 129 | 220 | | | | | |
| | Rating | 12 | 5 | 10 | 4 | 0 | 3 | 19 | 53 |

| Borehole | Footage | | RQD | Intact MPa Rock Strength | Joint Spacing | Separation of Joints | Continuity of Joints | Strike and Dip Orientation | Weathering & Groundwater | Score |
|---|---|---|---|---|---|---|---|---|---|---|
| 37934 | 2-13.5 | Description<br>Rating | 87%<br>14 | 76.4<br>5 | 191<br>10 | 5 | 3 | 5 | 19 | 61 |
| " | 13.5-18 | Description<br>Rating | 77%<br>14 | 5 | 220<br>10 | 5 | 3 | 5 | 19 | 61 |
| " | 18-25 | Description<br>Rating | 68%<br>12 | 5 | 97<br>10 | 5 | 3 | 5 | 19 | 59 |
| " | 25-29 | Description<br>Rating | 74%<br>12 | 5 | 172<br>10 | 5 | 3 | 5 | 19 | 59 |
| " | 29-33 | Description<br>Rating | 88%<br>7 | 5 | 175<br>10 | 5 | 3 | 5 | 19 | 54 |
| " | 33-36.5 | Description<br>Rating | 72%<br>12 | 5 | 153<br>10 | 5 | 3 | 5 | 19 | 59 |
| " | 36.5-50 | Description<br>Rating | 41%<br>[illegible] | 77<br>2 | 203<br>10 | 5 | 5 | 3 | 19 | 60 |
| " | 50-62.5 | Description<br>Rating | 66%<br>12 | 2 | 101<br>10 | 5 | 5 | 3 | 19 | 56 |
| " | 62.5-75 | Description<br>Rating | 83%<br>14 | 2 | 159<br>10 | 5 | 3 | 3 | 19 | 56 |
| " | 75-88 | Description<br>Rating | 67%<br>12 | 3 | 229<br>10 | 4 | 0 | 3 | 19 | 51 |
| " | 88-94 | Description<br>Rating | 74%<br>12 | 5 | 162<br>10 | 5 | 3 | 3 | 19 | 57 |

| Borehole Footage | | RQD | Intact MPa Rock Strength | Joint Spacing | Separation of Joints | Continuity of Joints | Strike and Dip Orientation | Weathering & Groundwater | Score |
|---|---|---|---|---|---|---|---|---|---|
| 37094 94-100 | Description | | | | | | | | |
| | Rating | 12 | 5 | 20 | 4 | 0 | 3 | 19 | 63 |
| " 100-104 | Description | 92% | | 352 | | | | | |
| | Rating | 16 | 5 | 20 | 5 | 5 | 3 | 19 | 73 |
| " 104-113.5 | Description | 96% | | 543 | | | | | |
| | Rating | 16 | 5 | 20 | 5 | 3 | 3 | 19 | 71 |
| " 113.5-121.1 | Description | 90% | 145 | 334 | | | | | |
| | Rating | 16 | 5 | 20 | 5 | 3 | 3 | 19 | 71 |
| " 121.1-125 | Description | 94% | | 153 | | | | | |
| | Rating | 14 | 5 | 10 | 4 | 3 | 3 | 19 | 58 |
| " 125-131 | Description | 88% | | 336 | | | | | |
| | Rating | 14 | 5 | 20 | 5 | 3 | 5 | 19 | 69 |

| Borehole | Footage | | | RQD | Intact MPa Rock Strength | Joint Spacing | Separation of Joints | Continuity of Joints | Strike and Dip Orientation | Weathering & Groundwater | Score |
|---|---|---|---|---|---|---|---|---|---|---|---|
| 35094 | 0-5 | Description | 75% | | | 71 | | | | | |
| | | Rating | 14 | 10 | | 25 | 5 | 5 | 5 | 19 | 83 |
| " | 5-13 | Description | 94% | | | 422 | | | | | |
| | | Rating | 16 | 10 | | 20 | 5 | 5 | 5 | 19 | 80 |
| " | 13-24 | Description | 82% | | 236 | 879 | | | | | |
| | | Rating | 14 | 10 | | 20 | 5 | 5 | 5 | 19 | 78 |
| " | 24-39 | Description | 87% | | | 359 | | | | | |
| | | Rating | 14 | 10 | | 20 | 5 | 5 | 5 | 19 | 78 |
| " | 39-45 | Description | 91% | | | 323 | | | | | |
| | | Rating | 16 | 8 | | 20 | 5 | 5 | 5 | 19 | 78 |
| " | 45-50 | Description | 93% | | | 359 | | | | | |
| | | Rating | 16 | 7 | | 20 | 5 | 5 | 5 | 19 | 77 |
| " | 50-56 | Description | 83% | | | 415 | | | | | |
| | | Rating | 14 | 5 | | 20 | 5 | 3 | 5 | 19 | 71 |
| " | 56-61 | Description | 80% | | | 269 | | | | | |
| | | Rating | 14 | 5 | | 10 | 5 | 5 | 5 | 19 | 63 |
| " | 61-67 | Description | 89% | | | 457 | | | | | |
| | | Rating | 14 | 5 | | 20 | 5 | 5 | 5 | 19 | 73 |
| " | 67-75 | Description | 93% | | 151 | 344 | | | | | |
| | | Rating | 16 | 5 | | 20 | 5 | 5 | 5 | 19 | 75 |
| " | 75-84.5 | Description | 82% | | | 233 | | | | | |
| | | Rating | 14 | 5 | | 10 | 5 | 5 | 5 | 19 | 63 |

| Borehole | Footage | | RQD | Intact MPa Rock Strength | Joint Spacing | Separation of Joints | Continuity of Joints | Strike and Dip Orientation | Weathering & Groundwater | Score |
|---|---|---|---|---|---|---|---|---|---|---|
| 35094 | 84.5–89 | Description<br>Rating | 82%<br>14 | 5 | 92<br>10 | 5 | 5 | 5 | 19 | 63 |
| " | 89–100 | Description<br>Rating | 63%<br>12 | 4 | 214<br>10 | 4 | 0 | 3 | 19 | 52 |
| " | 100–106 | Description<br>Rating | 91%<br>16 | 3 | 158<br>10 | 5 | 3 | 3 | 19 | 59 |
| " | 106–109 | Description<br>Rating | 52%<br>12 | 2 | 233<br>10 | 4 | 0 | 3 | 19 | 50 |
| " | 109–118 | Description<br>Rating | 73%<br>12 | 2 | 374<br>10 | 4 | 0 | 3 | 19 | 60 |
| " | 118–125 | Description<br>Rating | 88%<br>14 | 2 | 370<br>20 | 5 | 3 | 3 | 19 | 66 |
| " | 125–133 | Description<br>Rating | 89%<br>14 | 2 | 400<br>20 | 5 | 5 | 3 | 19 | 68 |
| " | 133–139 | Description<br>Rating | 89%<br>14 | 2 | 264<br>10 | 5 | 5 | 3 | 19 | 58 |
| " | 139–143 | Description<br>Rating | 79%<br>14 | 2 | 211<br>10 | 4 | 3 | 3 | 19 | 55 |
| " | 143–150 | Description<br>Rating | 90%<br>14 | 2 | 276<br>10 | 4 | 0 | 3 | 19 | 52 |

| Borehole Footage | | RQD | Intact MPa Rock Strength | Joint Spacing | Separation of Joints | Continuity of Joints | Strike and Dip Orientation | Weathering & Groundwater | Score |
|---|---|---|---|---|---|---|---|---|---|
| " 150-154 | Description | 91% | | 220 | | | | | |
| | Rating | 16 | 2 | 10 | 5 | 5 | 3 | 19 | 60 |
| 154-161 | Description | 92% | | 216 | | | | | |
| | Rating | 14 | 2 | 10 | 5 | 3 | 3 | 19 | 56 |
| " 161-164 | Description | 22% | | <50 mm | | | | | |
| | Rating | 3 | 2 | 5 | 5 | 3 | 3 | 19 | 40 |
| 164-174 | Description | 85% | | 212 | | | | | |
| | Rating | 14 | 2 | 10 | 5 | 3 | 3 | 19 | 56 |
| " 174-183 | Description | 74% | | 175 | | | | | |
| | Rating | 12 | 2 | 10 | 5 | 3 | 3 | 19 | 54 |
| 183-190 | Description | 63% | | 151 | | | | | |
| | Rating | 12 | 2 | 10 | 5 | 3 | 3 | 19 | 54 |
| " 190-200 | Description | 88% | 79.5% | 245 | | | | | |
| | Rating | 14 | 2 | 10 | 5 | 5 | 3 | 19 | 58 |
| 200-210 | Description | 94% | | 308 | | | | | |
| | Rating | 16 | 2 | 20 | 5 | 5 | 3 | 19 | 70 |
| " 210-219 | Description | 40% | | 140 | | | | | |
| | Rating | 7 | 2 | 10 | 5 | 3 | 3 | 19 | 49 |

| Borehole Footage | | RQD | Intact MPa Rock Strength | Joint Spacing | Separation of Joints | Continuity of Joints | Strike and Dip Orientation | Weathering & Groundwater | Score |
|---|---|---|---|---|---|---|---|---|---|
| 37395 1-11 | Rating | 93 16 | 136 5 | 333 20 | 5 | 5 | 5 | 19 | 75 |
| " 11-23 | Description Rating | 84% 14 | 136 5 | 517 20 | 5 | 5 | 5 | 19 | 73 |
| " 23-36 | Description Rating | 97% 16 | 5 | 152 10 | 5 | 3 | 5 | 19 | 63 |
| " 36-47 | Description Rating | 84% 14 | 5 | 182 10 | 5 | 3 | 5 | 19 | 61 |
| " 47-57 | Description Rating | 74% 12 | 5 | 169 10 | 5 | 3 | 5 | 19 | 59 |
| " 57-67 | Description Rating | 91% 16 | 5 | 250 10 | 5 | 3 | 5 | 19 | 63 |
| " 67-75 | Description Rating | 79% 14 | 97 5 | 192 10 | 5 | 3 | 5 | 19 | 61 |
| " 75-80.5 | Description Rating | 68% 12 | 5 | 135 10 | 5 | 3 | 5 | 19 | 59 |
| " 80.5-89 | Description Rating | 68% 12 | 5 | 141 10 | 5 | 3 | 5 | 19 | 59 |
| " 89-92 | Description Rating | 61% 12 | 5 | 158 10 | 5 | 3 | 5 | 19 | 59 |
| " 92-103 | Description Rating | 70% 12 | 5 | 338 20 | 5 | 3 | 5 | 19 | 69 |

| Borehole | Footage | | RQD | Intact MPa Rock Strength | Joint Spacing | Separation of Joints | Continuity of Joints | Strike and Dip Orientation | Weathering & Groundwater | Score |
|---|---|---|---|---|---|---|---|---|---|---|
| 35095 | 0-8.5 | Description<br>Rating | 83%<br>14 | 5 | 321<br>20 | 5 | 5 | 5 | 19 | 73 |
| | 8.5-16.5 | Description<br>Rating | 90%<br>16 | 5 | 700<br>20 | 5 | 5 | 5 | 19 | 75 |
| " | 16.5-22 | Description<br>Rating | 85%<br>14 | 5 | 140<br>10 | 5 | 3 | 5 | 19 | 61 |
| | 22-29.5 | Description<br>Rating | 73%<br>12 | 5 | 156<br>10 | 5 | 3 | 5 | 19 | 59 |
| " | 29.5-37 | Description<br>Rating | 90%<br>14 | 5 | 283<br>10 | 5 | 5 | 5 | 19 | 63 |
| | 37-42 | Description<br>Rating | 96%<br>16 | 184<br>5 | 440<br>20 | 5 | 3 | 5 | 19 | 73 |
| " | 42-50 | Description<br>Rating | 83%<br>14 | 5 | 302<br>20 | 5 | 3 | 5 | 19 | 71 |
| | 50-59 | Description<br>Rating | 84%<br>14 | 5 | 340<br>20 | 5 | 3 | 5 | 19 | 71 |
| " | 59-65 | Description<br>Rating | 89%<br>14 | 8 | 467<br>20 | 5 | 5 | 5 | 19 | 76 |

| Borehole Footage | | RQD | Intact MPa Rock Strength | Joint Spacing | Separation of Joints | Continuity of Joints | Strike and Dip Orientation | Weathering & Groundwater | Score |
|---|---|---|---|---|---|---|---|---|---|
| 35095 65-75 | Description | 76% | 357 | 318 | 5 | 5 | 5 | 19 | 78 |
| | Rating | 14 | 10 | 20 | | | | | |
| " 75-83 | Description | 32% | | 175 | 4 | 3 | 5 | 19 | 55 / 50 |
| | Rating | 7 | 7 | 10 | | | | | |
| " 83-89.5 | Description | 71% | 141 | | 4 | 0 | 3 | 19 | 50 |
| | Rating | 12 | 2 | 10 | | | | | |
| 89.5-94.5 | Description | 74% | 165 | | 4 | 3 | 3 | 19 | 53 |
| | Rating | 12 | 2 | 10 | | | | | |
| " 94.5-100 | Description | 99% | 119 | 726 | 5 | 5 | 3 | 19 | 73 |
| | Rating | 16 | 5 | 10 | | | | | |
| 100-105 | Description | 72% | | 233 | 5 | 3 | 3 | 19 | 64 |
| | Rating | 12 | 2 | 20 | | | | | |
| " 105-111 | Description | 67% | | 68 | 4 | 5 | 3 | 19 | 55 |
| | Rating | 12 | 2 | 10 | | | | | |
| 111-115.5 | Description | 61% | | 94 | 5 | 5 | 3 | 19 | 56 |
| | Rating | 12 | 2 | 10 | | | | | |
| " 115.5-125 | Description | 83% | 50.4 | 186 | 5 | 5 | 3 | 19 | 58 |
| | Rating | 14 | 2 | 10 | | | | | |

| Borehole | Footage | | RQD | Intact MPa Rock Strength | Joint Spacing | Separation of Joints | Continuity of Joints | Strike and Dip Orientation | Weathering & Groundwater | Score |
|---|---|---|---|---|---|---|---|---|---|---|
| 35095 | 125-130 | Description | 57% | | 156 | | | | | |
| | | Rating | 12 | 2 | 10 | 5 | 3 | 3 | 19 | 54 |
| " | 130-150 | Description | 70% | | 135 | | | | | |
| | | Rating | 12 | 2 | 10 | 5 | 5 | 3 | 19 | 56 |
| " | 150-154 | Description | 88% | | 352 | | | | | |
| | | Rating | 14 | 2 | 20 | 5 | 5 | 3 | 19 | 68 |
| | 154-169 | Description | 88% | | 312 | | | | | |
| | | Rating | 14 | 2 | 20 | 5 | 5 | 3 | 19 | 66 |
| " | 169-175 | Description | 91% | | 587 | | | | | |
| | | Rating | 16 | 5 | 20 | 5 | 5 | 3 | 19 | 73 |
| | 175-180 | Description | 87% | | 219 | | | | | |
| | | Rating | 14 | 5 | 10 | 5 | 5 | 3 | 19 | 61 |
| " | 180-188 | Description | 91% | 118 | 350 | | | | | |
| | | Rating | 14 | 5 | 20 | 5 | 5 | 3 | 19 | 71 |
| | 188-192 | Description | 44% | | 175 | | | | | |
| | | Rating | 7 | 5 | 10 | 4 | 0 | 3 | 19 | 48 |
| " | 192-200 | Description | 63% | | 200 | | | | | |
| | | Rating | 10 | 5 | 10 | 4 | 3 | 3 | 19 | 56 |
| " | 200-214 | Description | 94% | 179 | 204 | | | | | |
| | | Rating | 16 | 5 | 10 | 5 | 3 | 3 | 19 | 61 |

# APPENDIX E

# BLAST DESIGN DETAIL

| Hole | Total Length of Hole Broken (m) | Firing Sequence | Delay | Firing Time (m/sec) | Explosive Length (m) | Explosive Type | Charge Weight (kg) | Top of Plug From Hole Bottom (cm) | Length of Inter. Stemming (m) |
|---|---|---|---|---|---|---|---|---|---|
| 31E L | 7.0 | 1 | S.P. 0 | 0 | 2.0 | ANFO | 20.4 | 91 | 1.2 |
| U | | 2 | " 1 | 30 | " | " | " | | |
| 35F L | 6.7 | 2 | " 1 | 30 | " | " | " | 91 | 1.2 |
| U | | 3 | " 2 | 50 | " | " | " | | |
| 31F L | 7.3 | 3 | " 2 | 50 | " | " | " | 91 | 1.5 |
| U | | 4 | " 3 | 75 | " | " | " | | |
| 33I L | 6.7 | 4 | " 3 | 75 | " | " | " | 91 | 1.2 |
| U | | 5 | " 5 | 128 | " | " | " | | |
| 31J L | 7.3 | 5 | " 5 | 128 | " | " | " | 91 | 1.5 |
| U | | 6 | " 7 | 190 | " | " | " | | |
| 35G L | | 6 | " 7 | 190 | 1.5 | " | 15.3 | | 1.2 |
| M | 8.5 | 7 | " 9 | 280 | " | " | | 91 | 1.2 |
| U | | 8 | " 11 | 410 | " | " | | | |
| 33J L | | 7 | " 9 | 280 | " | " | " | | 1.2 |
| M | 8.2 | 8 | " 11 | 410 | " | " | " | 91 | 1.2 |
| U | | 9 | " 13 | 570 | " | " | " | | |
| 35H L | 9.5 | 10 | " 15 | 725 | 3.7 | POLYANFO | 13.6 | 61 | 1.2 |
| U | | 11 | " 18 | 950 | " | " | " | | |
| 34I L | 9.1 | 12 | " 21 | 1225 | " | " | " | 61 | 1.2 |
| U | | 13 | " 23 | 1675 | " | " | " | | |
| 33K L | 9.1 | 14 | " 25 | 2275 | " | " | " | 61 | 1.2 |
| U | | 15 | L.P. 9 | 3050 | " | " | " | | |
| 32K L | 9.1 | 16 | " 11 | 3900 | " | " | " | 61 | 1.2 |
| U | | 17 | " 13 | 4850 | " | " | " | | |
| 31K L | 8.5 | 18 | " 16 | 6550 | " | " | " | 61 | 1.2 |
| U | | 19 | " 18 | 8050 | " | " | " | | |

Location of deck in hole: U = upper deck
M = middle deck
L = lower deck

| Hole | Total Length of Hole Broken (m) | Firing Sequence | Delay | Firing Time (m/sec) | Explosive Length (m) | Explosive Type | Charge Weight (kg) | Top of Plug From Hole Bottom (cm) | Length of Inter. Stemming (m) |
|---|---|---|---|---|---|---|---|---|---|
| 31D L | 7.3 | 1 | S.P. 0 | 0 | 2.1 | ANFO | 12.8 | 91 | |
| U | | 2 | " 1 | 30 | " | " | " | | 1.2 |
| 35D L | | 3 | " 2 | 50 | 1.8 | " | 18.6 | | 1.2 |
| M | 9.1 | 4 | " 3 | 75 | " | " | " | 91 | |
| U | | 5 | 5 | 128 | " | | | | 1.2 |
| 33C L | | 6 | " 7 | 190 | " | " | " | | 1.2 |
| M | 9.8 | 7 | 9 | 280 | " | " | | 91 | |
| U | | 8 | " 11 | 410 | " | " | " | | 1.2 |
| 31C L | | 9 | " 13 | 570 | " | " | " | 91 | 1.2 |
| M | 9.1 | 10 | " 15 | 725 | " | " | " | | |
| U | | 11 | " 18 | 950 | " | " | " | | 1.2 |
| 35C L | | 12 | " 21 | 1225 | " | " | " | | 1.2 |
| UM | 12.2 | 13 | " 23 | 1675 | " | " | " | | |
| LM | | 14 | " 25 | 2275 | " | " | " | 91 | 1.2 |
| U | | 15 | L.P. 9 | 3050 | " | " | " | | 1.2 |
| 32B L | | 16 | " 11 | 3900 | " | AQUAMEX | 29.0 | | 1.2 |
| UM | 12.8 | 17 | " 13 | 4850 | " | " | " | 91 | |
| UM | | 18 | " 16 | 6550 | " | " | " | | 1.2 |
| U | | 19 | 18 | 8050 | " | " | " | | 1.2 |
| | | | | | | | | | 1.2 |

| Hole | | Total Length of Hole Broken (m) | Firing Sequence | Collar Delay | I.T.H. Delay | Firing Time (m/sec) | Explosive Length (m) | Explosive Type | Charge Weight (kg) | Top of Plug From Hole Bottom (cm) | Length of Inter. Stemming (m) | Length of Collar Stemming (m) |
|---|---|---|---|---|---|---|---|---|---|---|---|---|
| 31E | L | 8.2 | 1 | - | SP 0 | 0 | 2.6 | ANFO | 26.3 | 91 | 1.2 | .91 |
| | U | | 2 | - | 1 | 30 | " | " | " | | | |
| 31F | L | 8.2 | 3 | - | " 3 | 75 | " | " | " | " | 1.2 | .91 |
| | U | | 4 | - | 5 | 128 | " | " | " | | | |
| 35F | L | 8.2 | 5 | - | " 7 | 190 | " | " | " | " | 1.2 | .91 |
| | M | | 6 | - | " 9 | 280 | " | " | " | | | |
| 33I | L | 8.2 | 7 | - | " 11 | 410 | " | " | " | " | 1.2 | .91 |
| | U | | 8 | - | " 13 | 570 | " | " | " | | | |
| 31J | L | 8.2 | 9 | - | " 15 | 725 | " | " | " | " | 1.2 | .91 |
| | U | | 10 | - | " 18 | 950 | " | " | " | | | |
| 35G | L | 5.2 | 11 | - | " 21 | 1225 | 4.1 | " | 40.8 | " | - | 1.83 |
| | U | | 12 | | 23 | 1675 | " | " | " | | | |
| 35H | L | 13.4 | 13 | SP 25 | " 0 | 2275 | 4.9 | POLYANFO | 25.0 | " | 1.2 | 1.52 |
| | U | | 14 | " | 1 | 2350 | " | " | " | | | |
| 34I | L | 10.4 | 15 | " | 3 | 2350 | 4.0 | " | 20.4 | 61 | 1.2 | .61 |
| | U | | 16 | | 5 | 2403 | " | " | " | | | |
| 33K | L | 10.4 | 17 | " | " 7 | 2465 | " | " | " | " | 1.2 | .61 |
| | U | | 18 | | 9 | 2555 | " | " | " | | | |
| 32K | L | 9.5 | 19 | " | " 11 | 2685 | 3.5 | " | 18.1 | " | 1.2 | .61 |
| | U | | 20 | | 13 | 2845 | " | " | " | | | |

| Hole | | Total Length of Hole Broken (m) | Firing Sequence | Collar Delay | I.T.H. Delay | Firing Time (m/sec) | Explosive Length (m) | Explosive Type | Charge Weight (kg) | Top of Plug From Hole Bottom (cm) | Length of Inter. Stemming (m) | Length of Collar Stemming (cm) |
|---|---|---|---|---|---|---|---|---|---|---|---|---|
| 31K | L | 9.5 | 21 | SP 25 | SP 15 | 3000 | 3.5 | POLYANFO | 18.1 | 61 | 1.2 | 61 |
| | U | | 22 | " | 18 | 3225 | " | " | " | | | |
| 31D | L | 8.2 | 23 | LP 12 | " 0 | 4350 | 2.1 | ANFO | 21.8 | 91 | 1.2 | 91 |
| | U | | 24 | " | 1 | 4380 | " | " | " | | | |
| 35D | L | 8.2 | 25 | " | " 3 | 4425 | " | " | " | " | 1.2 | 91 |
| | U | | 26 | | " 5 | 4478 | " | " | " | | | |
| 33C | L | 8.2 | 27 | " | " 7 | 4540 | " | " | " | " | 1.2 | 91 |
| | U | | 28 | | " 9 | 4630 | " | " | " | | | |
| 31C | L | 8.2 | 29 | " | " 11 | 4760 | " | " | " | " | 1.2 | 91 |
| | U | | 30 | | " 13 | 4920 | " | " | " | | | |
| 35C | L | 8.2 | 31 | " | " 15 | 5075 | " | AQUAMEX | 34.0 | " | 1.2 | 91 |
| | U | | 32 | | 18 | 5300 | " | " | " | | | |
| 32B | L | 8.2 | 33 | " | " 21 | 5575 | " | " | " | " | 1.2 | 91 |
| | U | | 34 | " | 23 | 6025 | " | " | " | | | |
| 34B | L | 8.2 | 35 | " | 25 | 6625 | " | " | " | " | 1.2 | 91 |
| | U | | 36 | | LP 9 | 7400 | " | " | " | | | |
| 31A | L | 8.2 | 37 | " | " 11 | 8250 | " | " | " | " | 1.2 | 91 |
| | U | | 38 | | 13 | 9200 | " | " | " | | | |
| 33A | L | 8.2 | 39 | " | " 16 | 10900 | " | " | " | " | 1.2 | 91 |
| | U | | 40 | " | " 18 | 12400 | " | " | " | " | | |

| Hole | | Total Length of Hole Broken (m) | Firing Sequence | Collar Delay | I.T.H. Delay | Firing Time (MS) | Explosive Length (m) | Explosive Type | Charge Weight (kg) | Top of Plug From Hole Bottom (m) | Length of Inter. Stemming (m) |
|---|---|---|---|---|---|---|---|---|---|---|---|
| 29N | L | | 1 | - | SP 1 | 30 | 3.4 | ANFO | 34.9 | 1 | 1.2 |
| | M | 15.2 | 2 | - | " 2 | 50 | " | " | " | | 1.2 |
| | U | | 3 | - | 3 | 75 | " | | " | | |
| 29P | L | | 4 | - | " 5 | 128 | " | " | " | 1 | 1.2 |
| | M | 15.2 | 5 | - | 7 | 190 | " | | | | 1.2 |
| | U | | 6 | - | " 9 | 280 | " | " | " | | |
| 29R | L | | 7 | - | " 11 | 410 | 3.7 | " | 37.9 | | 1.2 |
| | M | 15.9 | 8 | - | " 13 | 570 | " | " | " | 1 | 1.2 |
| | U | | 9 | - | " 15 | 725 | " | " | " | | |
| 30S | L | | 10 | SP 18 | SP 1 | 980 | 3.0 | POLYANFO | 16.9 | | 1.2 |
| | LM | 18.0 | 11 | " | " 2 | 1000 | " | " | " | 0.6 | 1.2 |
| | UM | | 12 | " | " 3 | 1025 | " | " | " | | 1.2 |
| | U | | 13 | " | " 5 | 1079 | " | | | | |
| 29S | L | | 14 | " | " 7 | 1140 | " | " | " | | 1.2 |
| | LM | 18.0 | 15 | " | " 9 | 1230 | " | " | " | 0.6 | 1.2 |
| | UM | | 16 | " | " 11 | 1360 | " | " | " | | 1.2 |
| | U | | 17 | " | " 13 | 1520 | " | | | | |
| 27P | L | | 18 | SP 24 | SP 1 | 1980 | 3.4 | ANFO | 34.9 | | 1.2 |
| | M | 15.2 | 19 | " | " 2 | 2000 | " | " | " | 1 | 1.2 |
| | U | | 20 | " | " 3 | 2025 | " | | | | |
| 27N | L | | 21 | " | " 5 | 2078 | " | " | " | | 1.2 |
| | M | 15.2 | 22 | " | " 7 | 2140 | " | " | " | 1 | 1.2 |
| | U | | 23 | " | " 9 | 2230 | " | | | | |
| 27R | L | | 24 | " | " 11 | 2360 | 3.7 | " | 37.9 | | 1.2 |
| | M | 15.9 | 25 | " | " 13 | 2520 | " | " | " | 1 | 1.2 |
| | U | | 26 | " | " 15 | 2675 | " | " | " | | |
| 28S | L | | 27 | UP 10 | SP 1 | 3480 | 3.0 | POLYANFO | 16.9 m | 0.6 | 1.2 |
| | LM | 18.0 | 28 | " | " 2 | 3500 | " | " | " | | 1.2 |
| | UM | | 29 | " | " 3 | 3525 | " | " | " | | 1.2 |
| | U | | 30 | " | " 5 | 3578 | " | " | " | | |

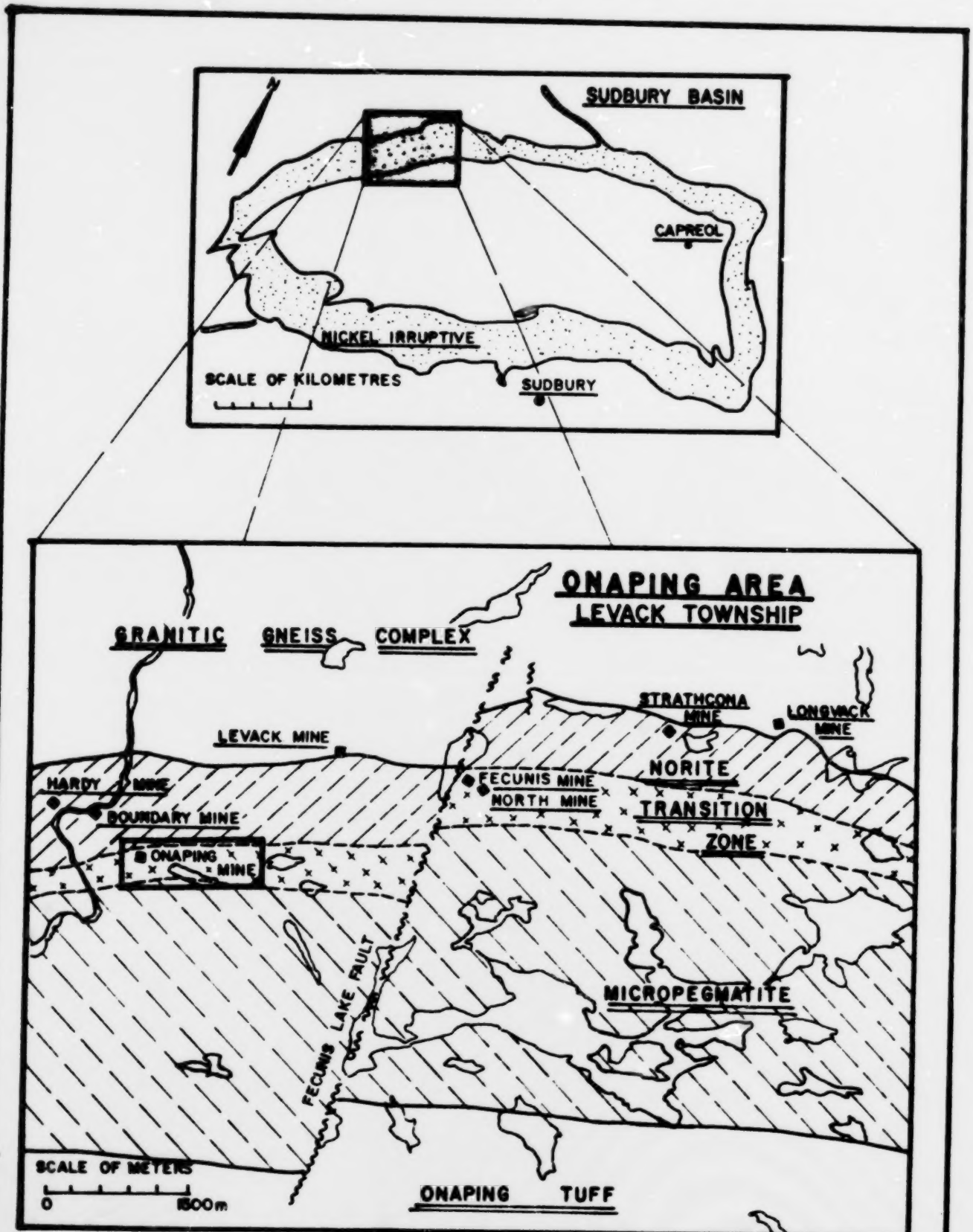
| Hole | Total Length of Hole Broken (m) | Firing Sequence | Collar Delay | I.T.H. Delay | Firing Time (MS) | Explosive Length (m) | Explosive Type | Charge Weight (kg) | Top of Plug From Hole Bottom (m) | Length of Inter. Stemming (m) |
|---|---|---|---|---|---|---|---|---|---|---|
| 27S L | 18.0 | 31 | LP 10 | " 7 | 3640 | 3.0 | PULf/ANFO | 16.9 | 0.6 | 1.2 |
| LM | | 32 | " | " 9 | 3730 | " | " | " | " | 1.2 |
| MU | | 33 | " | 11 | 3960 | " | " | " | " | |
| U | | 34 | " | 13 | 4020 | " | " | " | " | |
| 29M L | 15.9 | 35 | " | " 15 | 4175 | 3.7 | ANFO | 37.9 | 1 | 1.2 |
| M | | 36 | " | " 18 | 4400 | " | " | " | " | 1.2 |
| U | | 37 | " | 21 | 4675 | " | " | " | " | |

| Hole | | Total Length of Hole Broken (m) | Firing Sequence | Collar Delay | I.T.H. Delay | Firing Time (MS) | Explosive Length (m) | Explosive Type | Charge Weight (kg) | Top of Plug From Hole Bottom (m) | Length of Inter. Stemming (m) |
|---|---|---|---|---|---|---|---|---|---|---|---|
| 27M | L | | 1 | - | SP 1 | 30 | 3.6 m | ANFO | 36.9 | 1 | 1.2 |
| | M | 15.9 | 2 | - | " 2 | 50 | " | " | " | " | 1.2 |
| | U | | 3 | - | " 3 | 75 | " | " | " | " | |
| 29L | L | | 4 | - | " 5 | 128 | 3.4 m | " | 34.8 | " | 1.2 |
| | LM | 19.5 | 5 | - | " 7 | 190 | " | " | " | " | 1.2 |
| | UM | | 6 | - | " 9 | 280 | " | " | " | " | 1.2 |
| | U | | 7 | - | " 11 | 410 | " | " | " | " | |
| 27L | L | | 8 | - | SP 13 | 570 | " | " | " | " | 1.2 |
| | LM | 19.5 | 9 | - | " 15 | 725 | " | " | " | " | 1.2 |
| | UM | | 10 | - | " 18 | 950 | " | " | " | " | 1.2 |
| | U | | 11 | - | " 21 | 1225 | " | " | " | " | |
| 29A | L | | 12 | SP 23 | SP 1 | 1705 | 1.6 m | AQUAMEX | 25.6 | " | 1.2 |
| | LM | 15.2 | 13 | " | " 2 | 1725 | " | " | " | " | 1.2 |
| | M | | 14 | " | " 3 | 1750 | " | " | " | " | 1.2 |
| | UM | | 15 | " | " 5 | 1803 | " | " | " | " | 1.2 |
| | U | | 16 | " | " 7 | 1865 | " | " | " | " | |
| 27A | L | | 17 | " | SP 9 | 1955 | " | " | " | " | 1.2 |
| | LM | 15.2 | 18 | " | " 11 | 2085 | " | " | " | " | 1.2 |
| | M | | 19 | " | " 13 | 2245 | " | " | " | " | 1.2 |
| | UM | | 20 | " | " 15 | 2400 | " | " | " | " | 1.2 |
| | U | | 21 | " | " 18 | 2625 | " | " | " | " | |

Holes 29A and 27A should have a separate trunkline in addition to the regular hookup to avoid premature firing due to cap scatter. Both holes should be stemmed with sand to the overcut level.

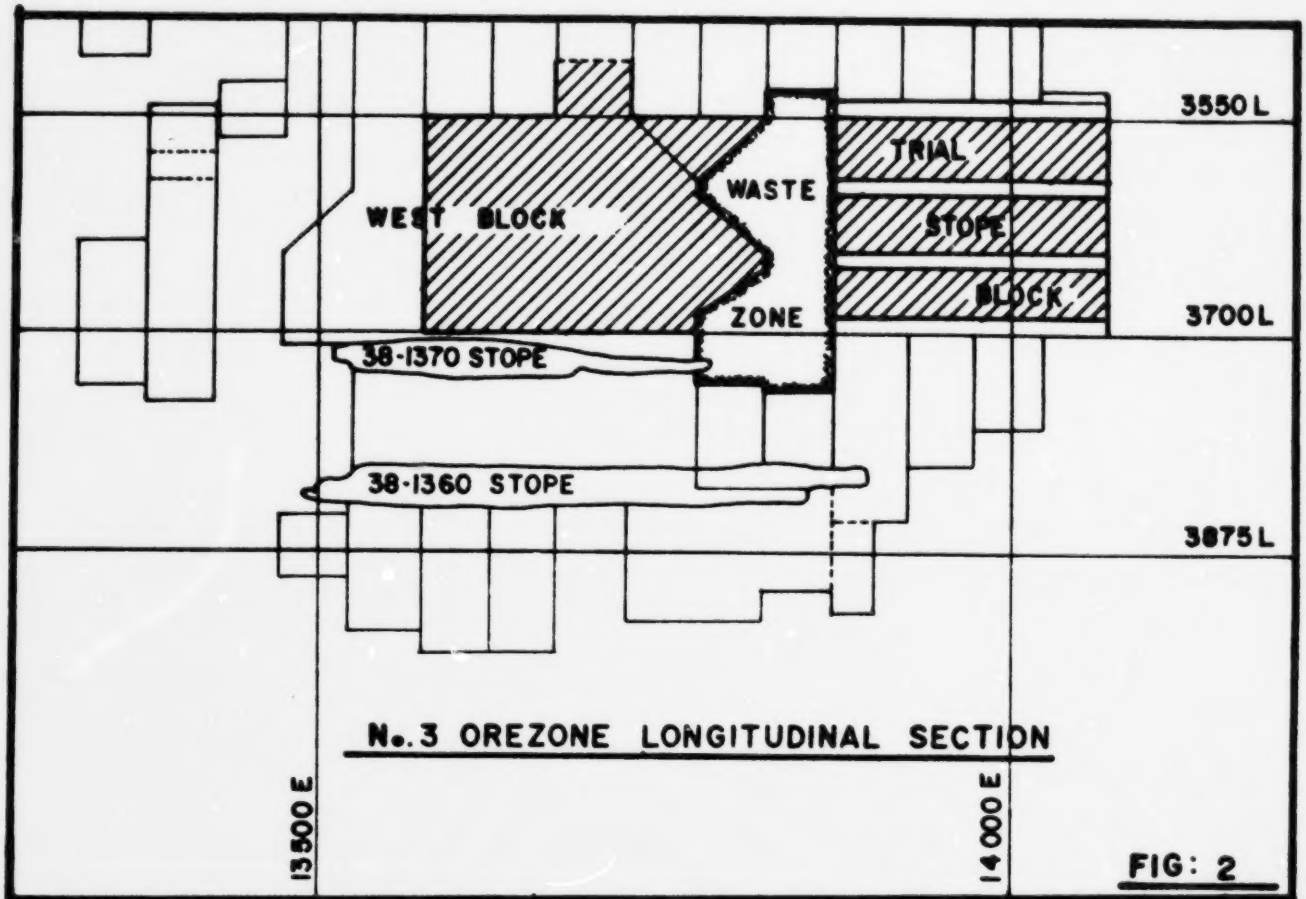
| Hole | | Total Length of Hole Broken (m) | Firing Sequence | Collar Delay | I.T.H. Delay | Firing Time (MS) | Explosive Length (m) | Explosive Type | Charge Weight (kg) | Top of Plug From Hole Bottom (m) | Length of Inter. Stemming (m) |
|---|---|---|---|---|---|---|---|---|---|---|---|
| 26G | L | | 1 | - | SP 1 | 30 | 4 | ANFO | 41 | 1 | 1.2 |
| | U | 12.5 | 2 | - | " 2 | 50 | " | " | " | | |
| 26H | L | | 3 | - | " 3 | 75 | " | " | " | 1 | 1.2 |
| | M | 14.6 | 4 | - | " 5 | 128 | " | " | " | | 1.2 |
| | U | | 5 | - | " 7 | 190 | 1.4 | " | 14.4 | | |
| 261 | L | | 6 | - | " 9 | 280 | 4 | POLYANFO | 20.4 | 0.6 | 1.2 |
| | M | 17.8 | 7 | - | " 11 | 410 | " | " | " | | 1.2 |
| | U | | 8 | - | " 13 | 570 | " | " | " | | |
| 26F | L | | 9 | - | SP 15 | 725 | " | ANFO | 41 | 1 | 1.2 |
| | M | 15.2 | 10 | - | " 18 | 950 | " | " | " | | 1.2 |
| | U | | 11 | - | " 21 | 1225 | " | " | " | | |
| 251 | L | | 12 | SP 23 | " 1 | 1705 | " | POLYANFO | 20.4 | 0.6 | 1.2 |
| | M | 17.8 | 13 | " | " 2 | 1725 | " | " | " | | 1.2 |
| | U | | 14 | " | " 3 | 1750 | " | " | " | | |
| 24G | L | 12.5 | 15 | SP 25 | SP 1 | 2305 | " | ANFO | 41 | 1 | 1.2 |
| | U | | 16 | " | " 2 | 2325 | " | " | " | | |
| | | | | " | | | | | | | |
| 24H | L | 14.6 | 17 | LP 9 | " 1 | 3080 | " | " | " | 1 | 1.2 |
| | M | | 18 | " | " 2 | 3100 | " | " | " | | 1.2 |
| | U | | 19 | " | " 3 | 3125 | 1.4 | " | 14.4 | | |
| 241 | L | 17.8 | 20 | LP 11 | " 1 | 3930 | 4 | POLYANFO | 20.4 | 0.6 | 1.2 |
| | M | | 21 | " | " 2 | 3950 | " | " | " | | 1.2 |
| | U | | 22 | " | " 3 | 3975 | " | " | " | | |
| 24F | L | 15.2 | 23 | LP 13 | SP 1 | 4880 | " | ANFO | 41 | 1 | 1.2 |
| | M | | 24 | " | " 2 | 4900 | " | " | " | | 1.2 |
| | U | | 25 | " | " 3 | 4925 | " | " | " | | |

| Hole | | Total Length of Hole Broken (m) | Firing Sequence | Collar Delay | I.T.H. Delay | Firing Time (MS) | Explosive Length (m) | Explosive Type | Charge Weight (kg) | Length of Inter. Stemming (m) |
|---|---|---|---|---|---|---|---|---|---|---|
| 26E | L | | 1 | - | SP 1 | 30 | 2 | AQUAMEX | 32 | 2 |
| | U | 11.3 | 2 | - | " 2 | 50 | " | " | " | 2 |
| | M | | 3 | - | " 3 | 75 | 1 | " | 16 | |
| 26D | L | | 4 | - | " 5 | 125 | 2.5 | " | 40 | 2 |
| | U | 9.1 | 5 | - | " 7 | 190 | " | " | " | |
| 26B | L | | 6 | - | " 9 | 290 | 2 | " | 32 | 2 |
| | M | 11.3 | 7 | - | " 11 | 410 | " | " | " | 2 |
| | U | | 8 | - | " 13 | 570 | 1 | " | 16 | |
| 26C | L | 7.3 | 9 | - | " 15 | 725 | 2 | " | 32 | 2 |
| | U | | 10 | - | " 18 | 950 | 1 | " | 16 | |
| 26D | L | 9.1 | 11 | - | " 21 | 1225 | 2.5 | " | 40 | 2 |
| | U | | 12 | - | " 23 | 1675 | " | " | " | |
| 26B | | 4.6 | 13 | - | " 25 | 2275 | " | " | 40 | |
| 24C | L | 7.3 | 14 | LP 9 | " 1 | 3080 | 2 | " | 32 | 2 |
| | U | | 15 | " | " 2 | 3100 | 1 | " | 16 | |
| 26A | | 3 | 16 | " | LP 13 | 4950 | 1 | " | 16 | |
| 24B | | 4.6 | 17 | " | " 16 | 6550 | 2.5 | " | 40 | |
| 24A | | 3 | 18 | " | " 18 | 8050 | 1 | " | 16 | |



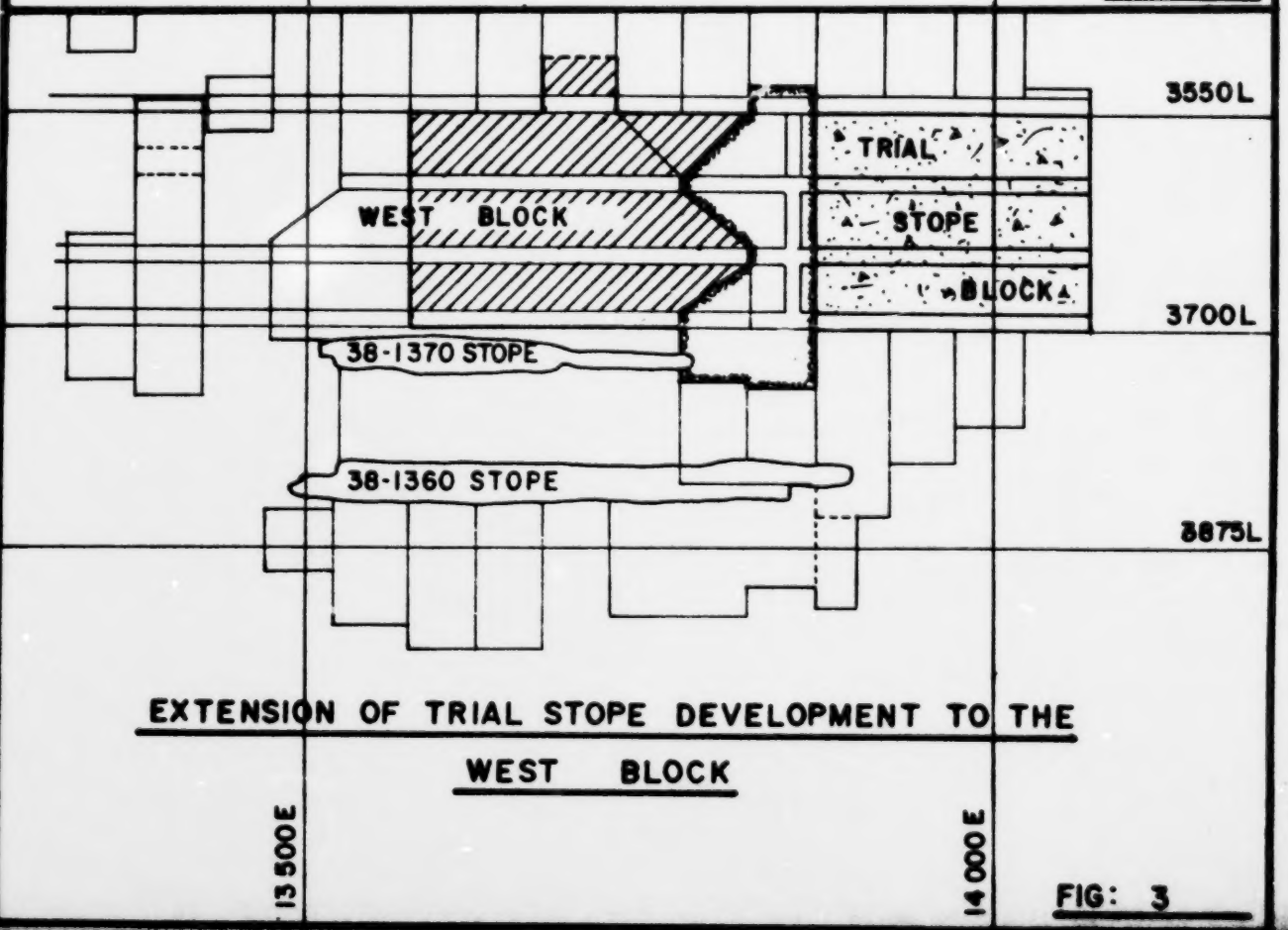

ONAPING MINE SURFACE LOCATION

FIG: 1



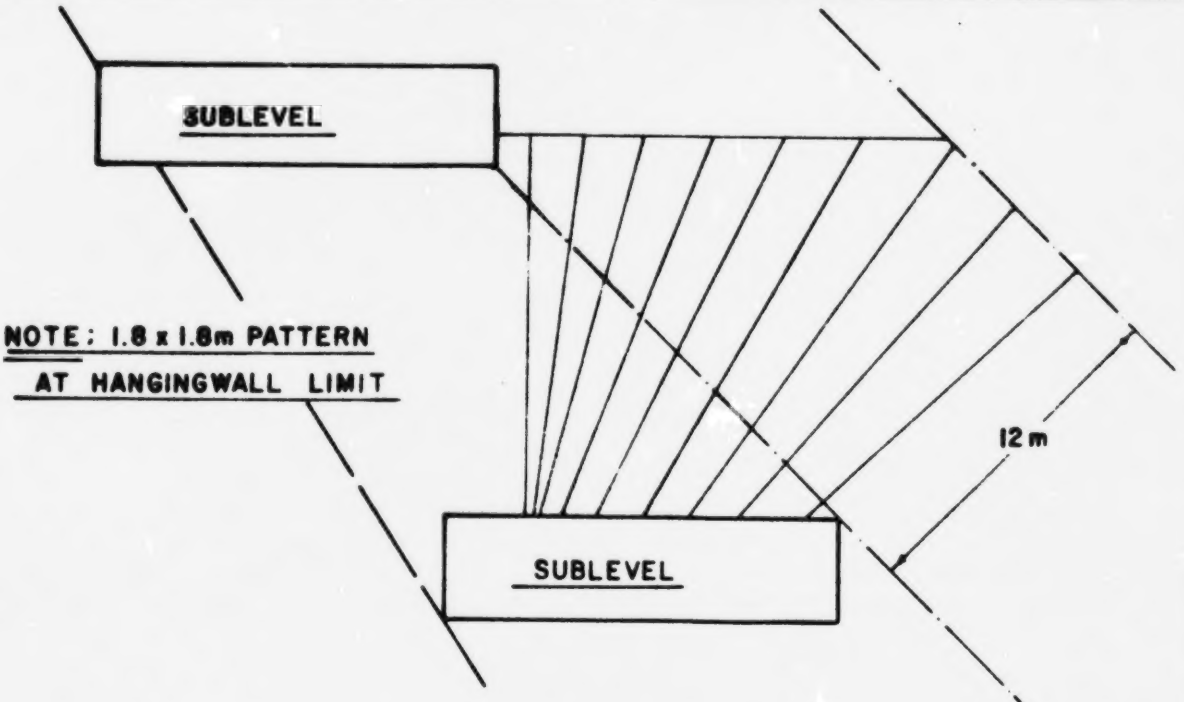

No. 3 OREZONE LONGITUDINAL SECTION

FIG: 2



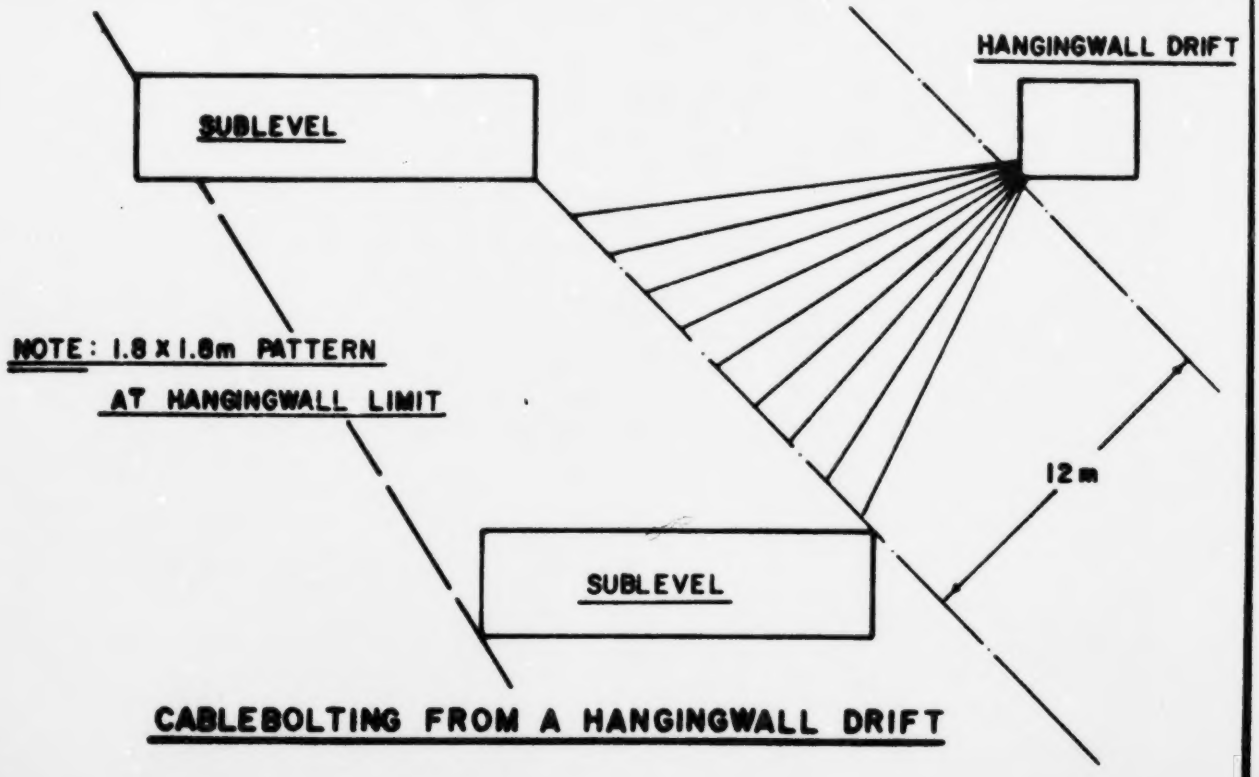

EXTENSION OF TRIAL STOPE DEVELOPMENT TO THE WEST BLOCK

FIG: 3




CABLEBOLTING FROM AN UNDERCUT

FIG: 4




CABLEBOLTING FROM A HANGINGWALL DRIFT

FIG: 5

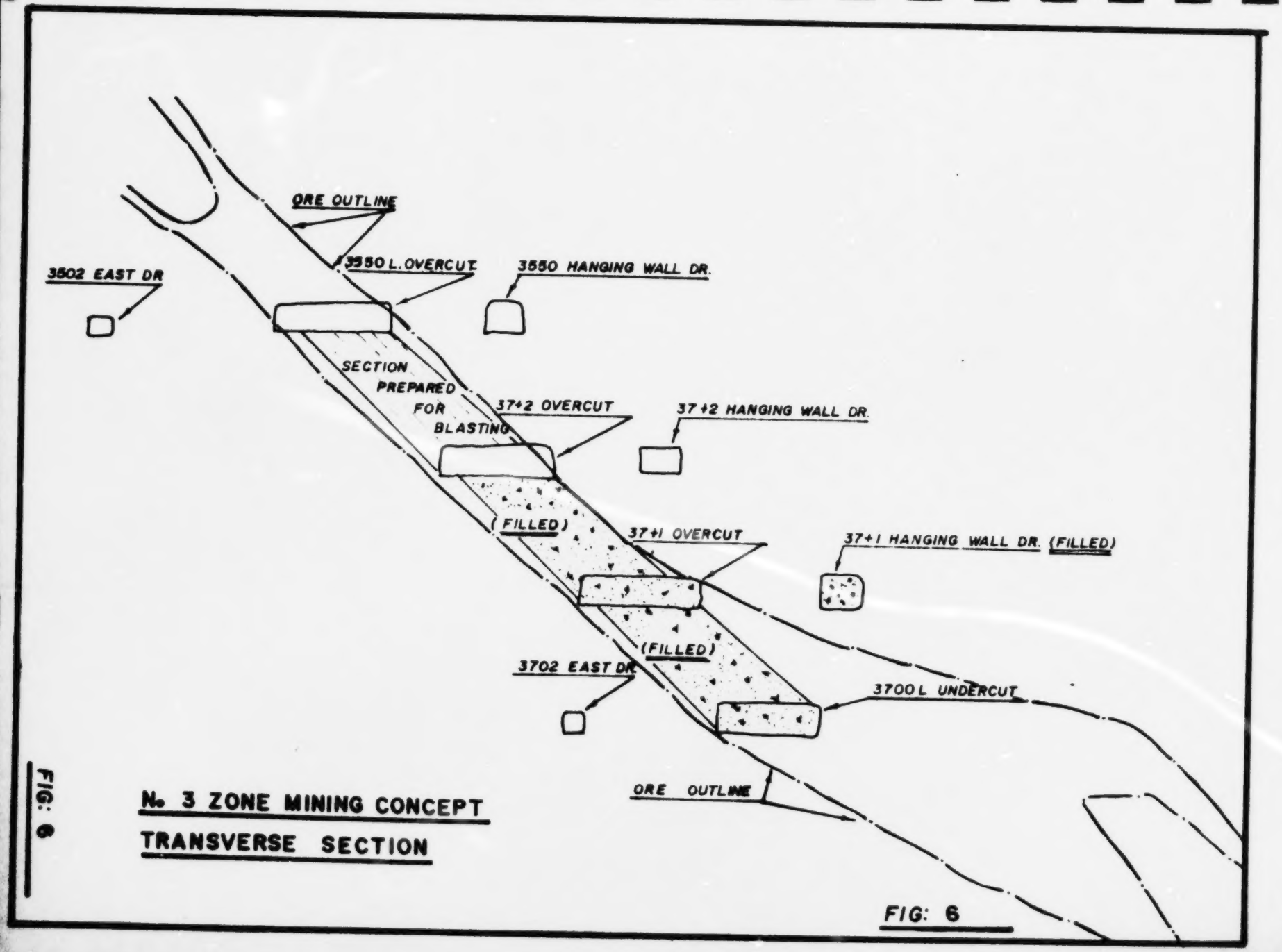



FIG: 6

No. 3 ZONE MINING CONCEPT
TRANSVERSE SECTION

FIG: 6

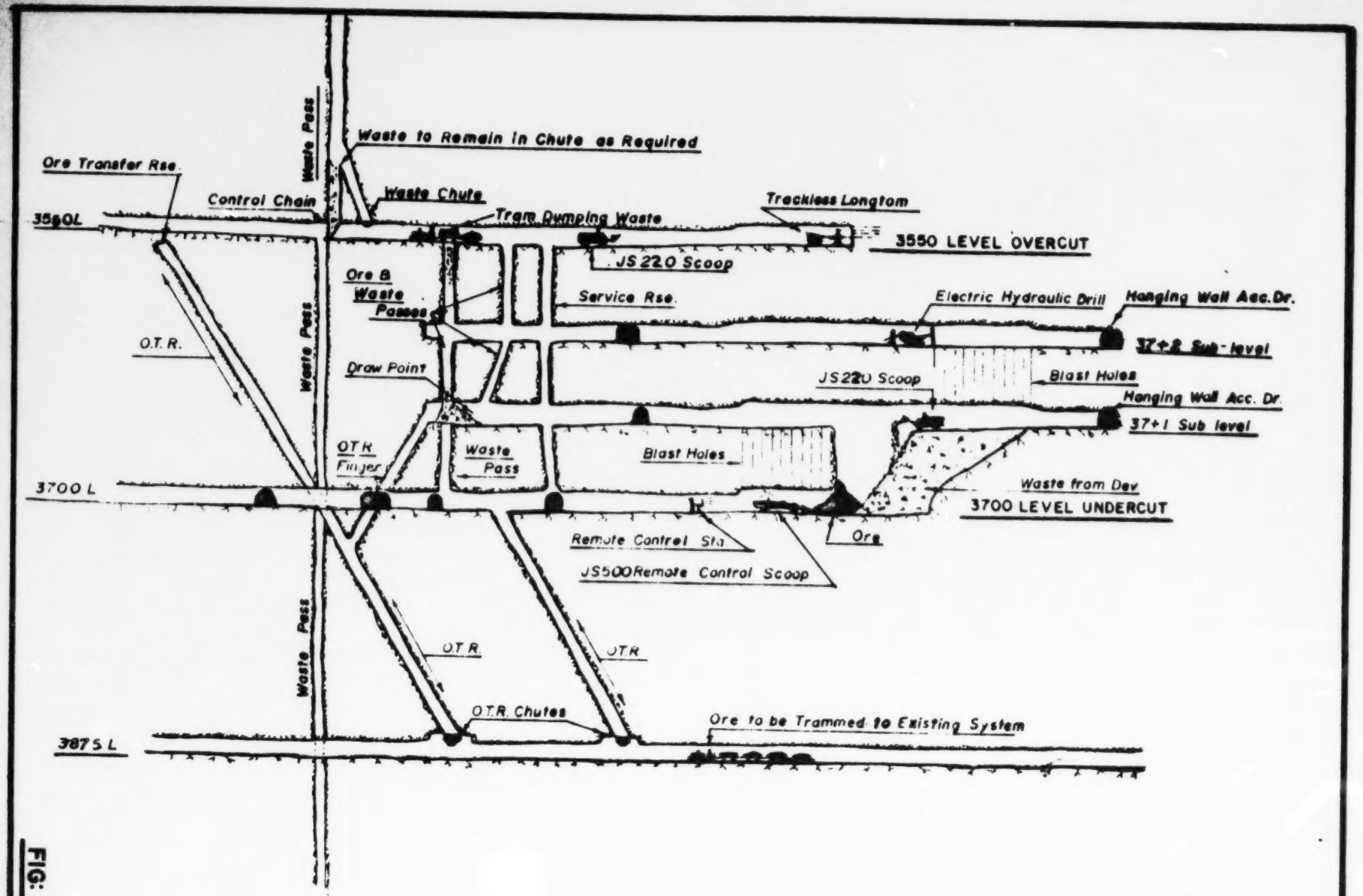



FIG: 7

## No. 3 ZONE LONGITUDINAL MINING CONCEPT

FIG: 7

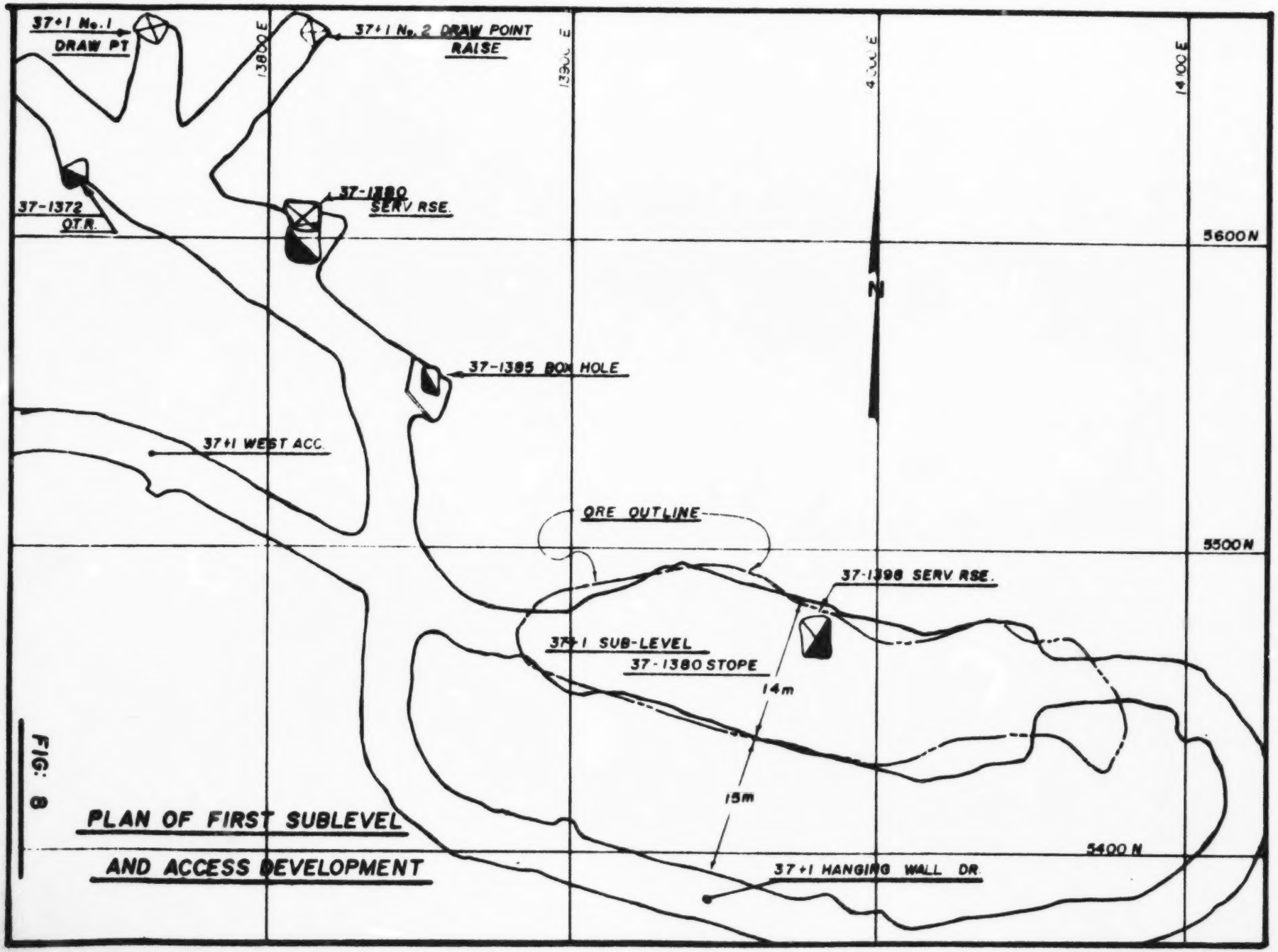
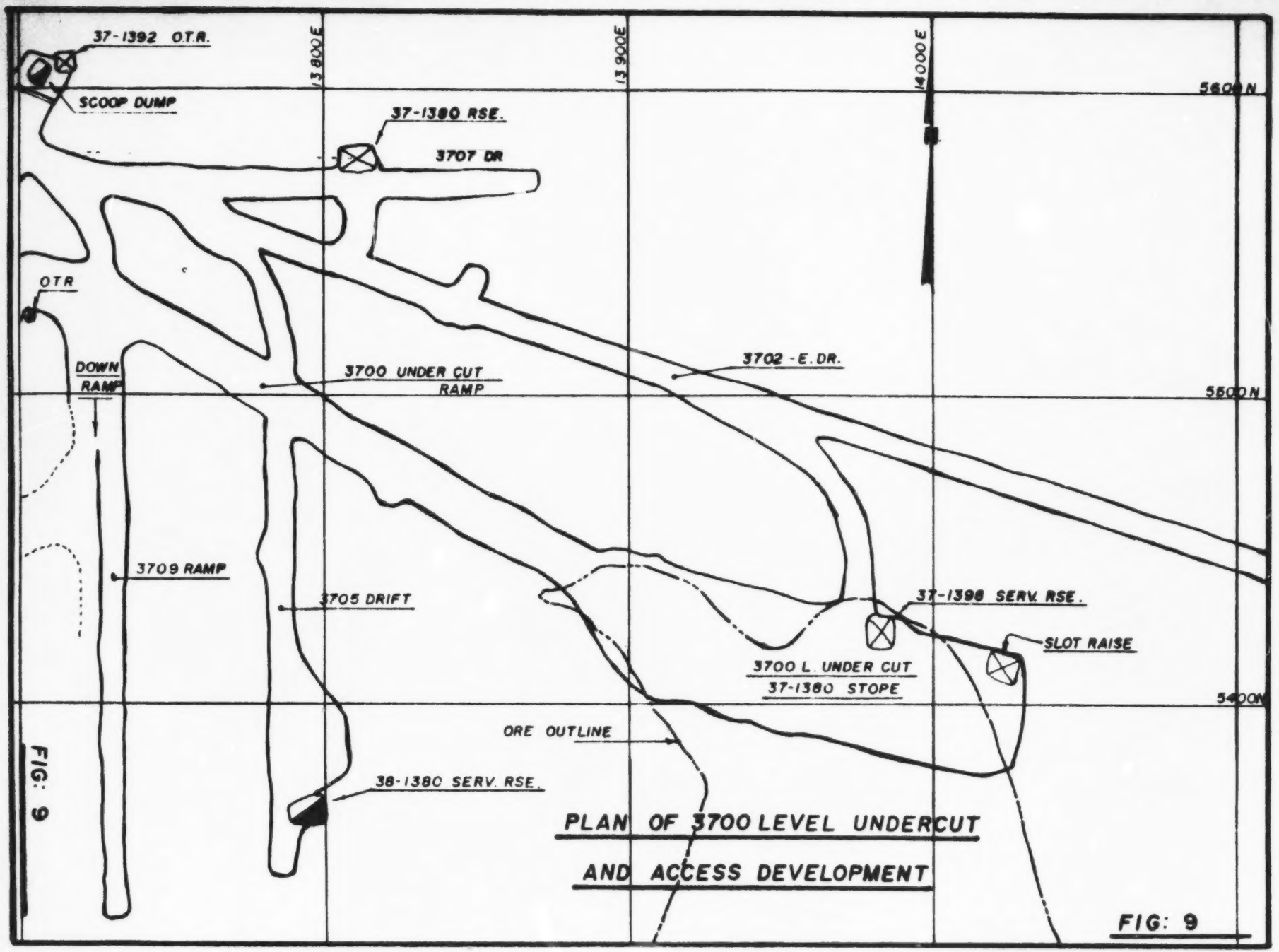
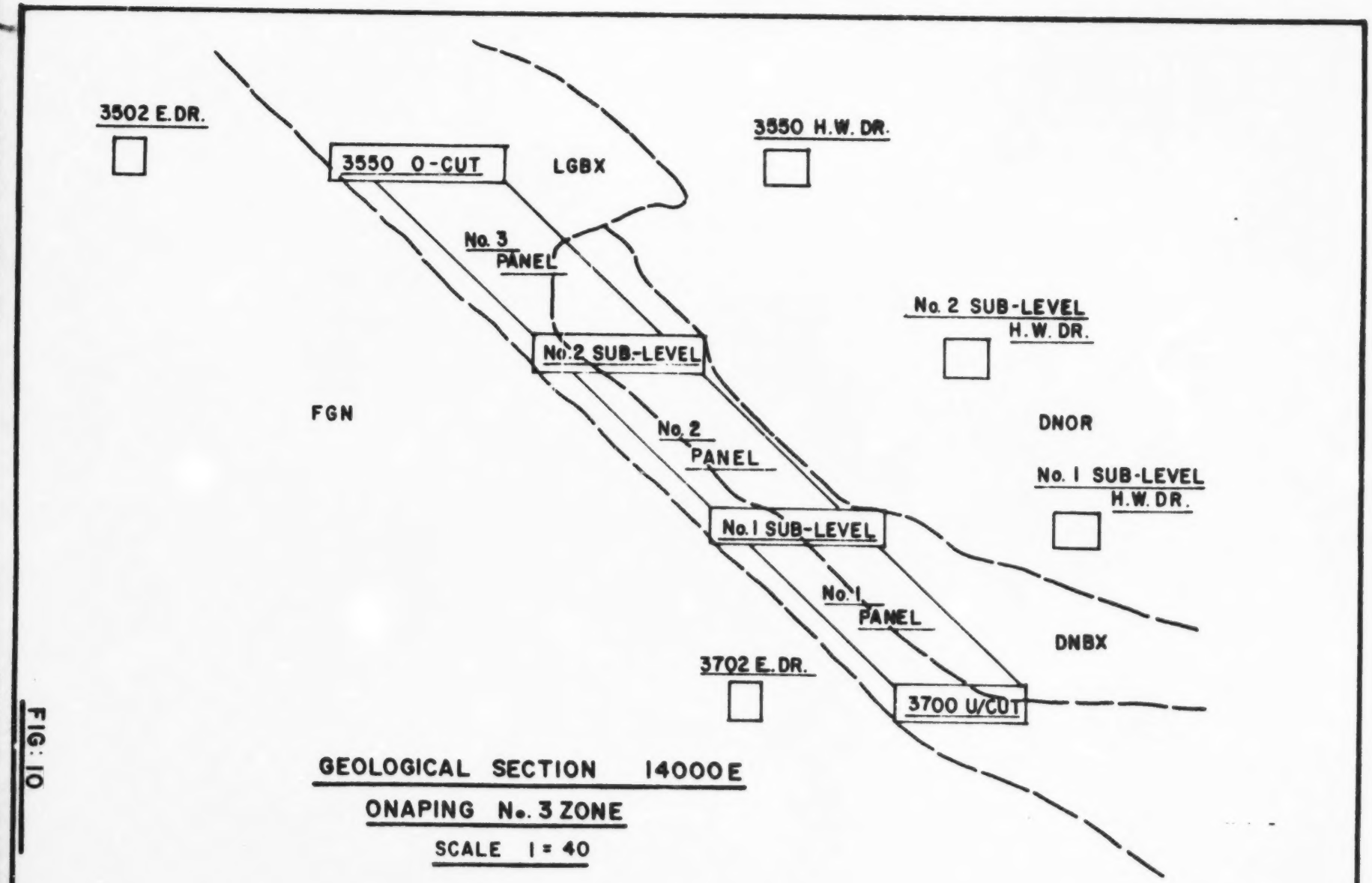



FIG: 8

PLAN OF FIRST SUBLEVEL
AND ACCESS DEVELOPMENT


37-1392 O.T.R.
SCOOP DUMP
13800E
13900E
14000E
5600N
37-1380 RSE.
3707 DR
OTR
DOWN RAMP
3700 UNDER CUT RAMP
3702 - E. DR.
5500N
3709 RAMP
3705 DRIFT
37-1398 SERV. RSE.
SLOT RAISE
3700 L. UNDER CUT
37-1380 STOPE
5400N
ORE OUTLINE
38-1380 SERV. RSE.
FIG: 9
PLAN OF 3700 LEVEL UNDERCUT
AND ACCESS DEVELOPMENT
FIG: 9



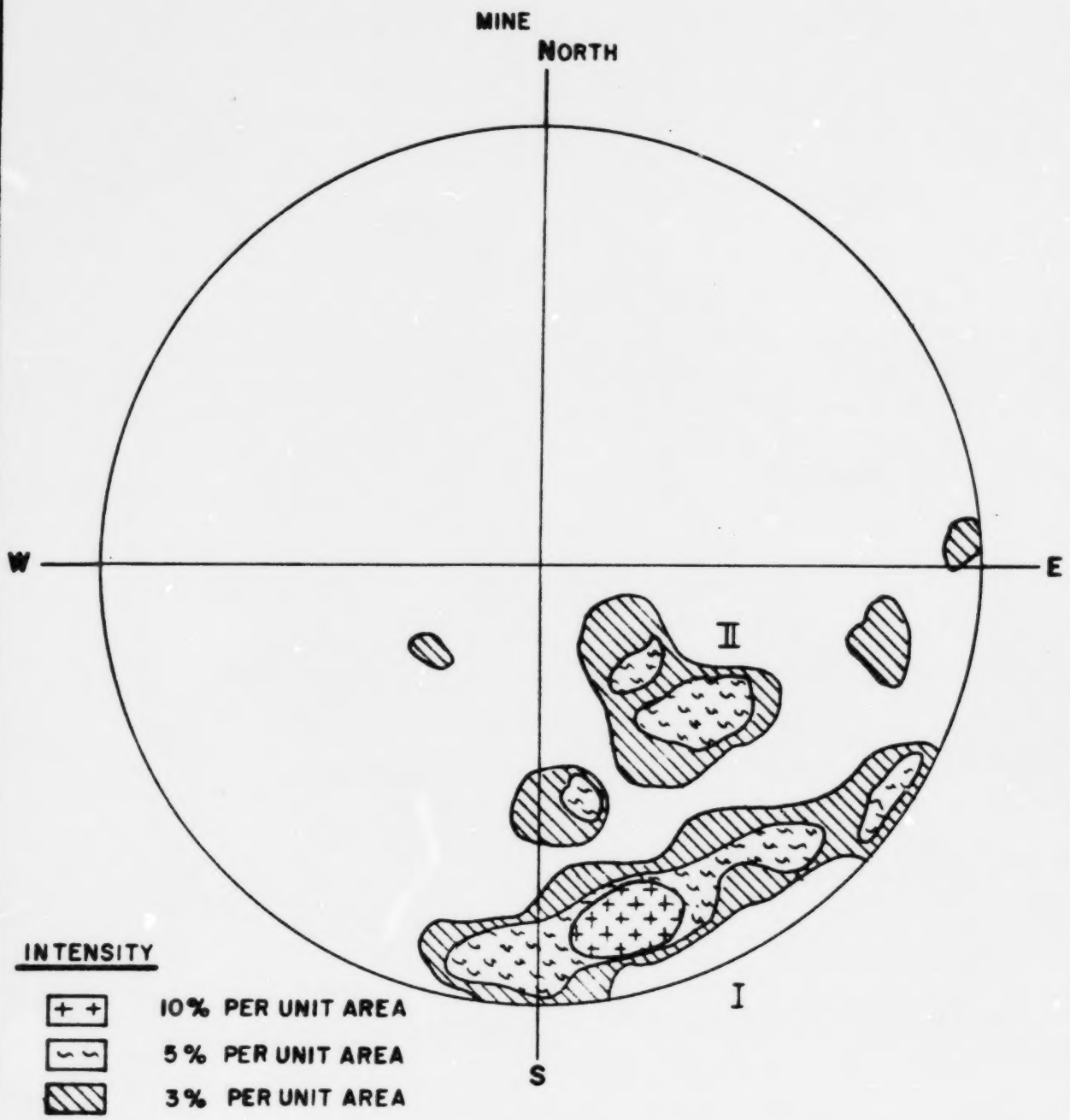

GEOLOGICAL SECTION 14000E

ONAPING No. 3 ZONE

SCALE 1 = 40

FIG: 10

FIG: 10

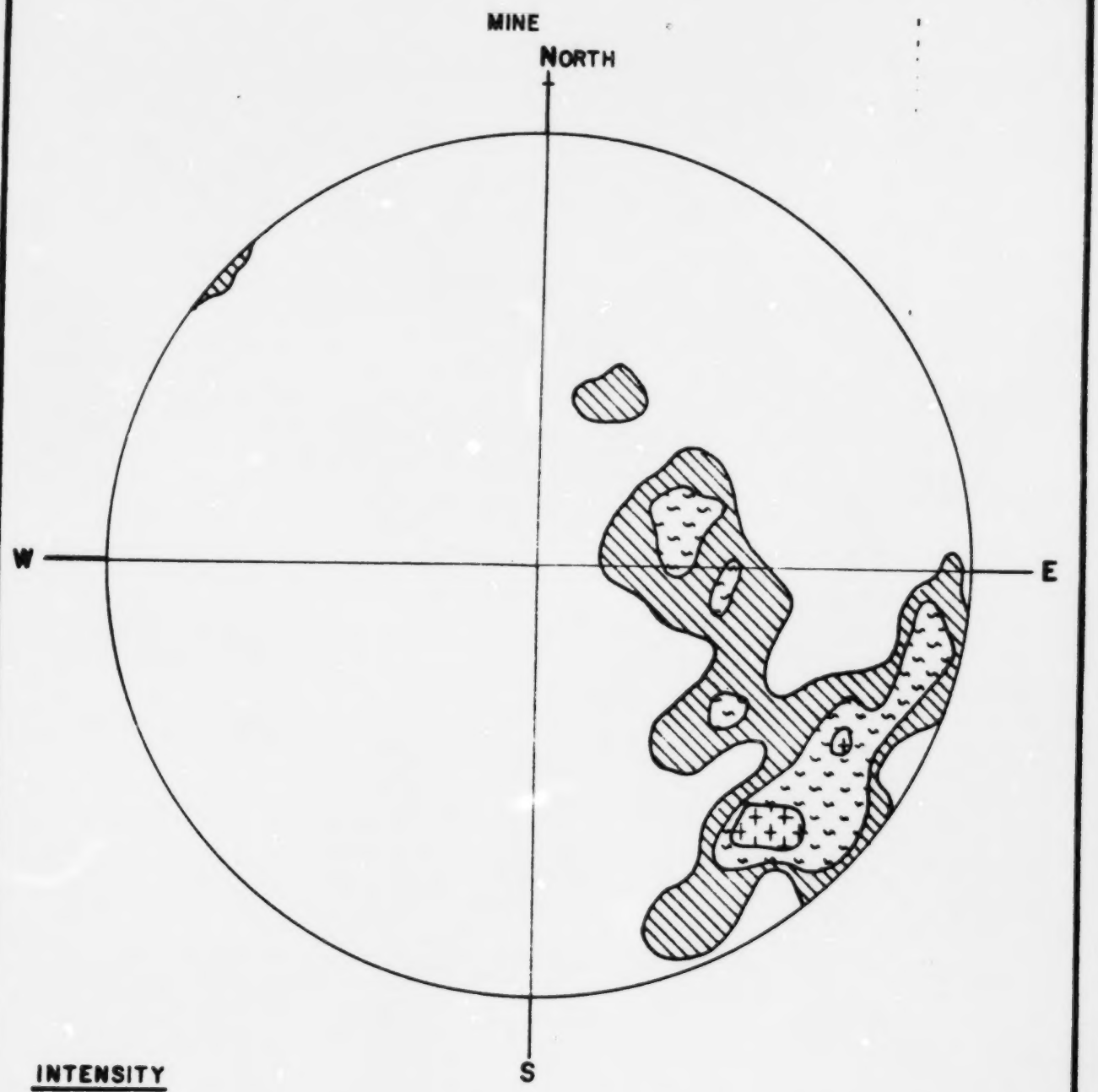



LOWER HEMISPHERE STEREOGRAPHIC PROJECTION

POLE TO JOINTING No. 3 ZONE FELDSPATHIC GNEISS

96 DATA POINTS

FIG: 11



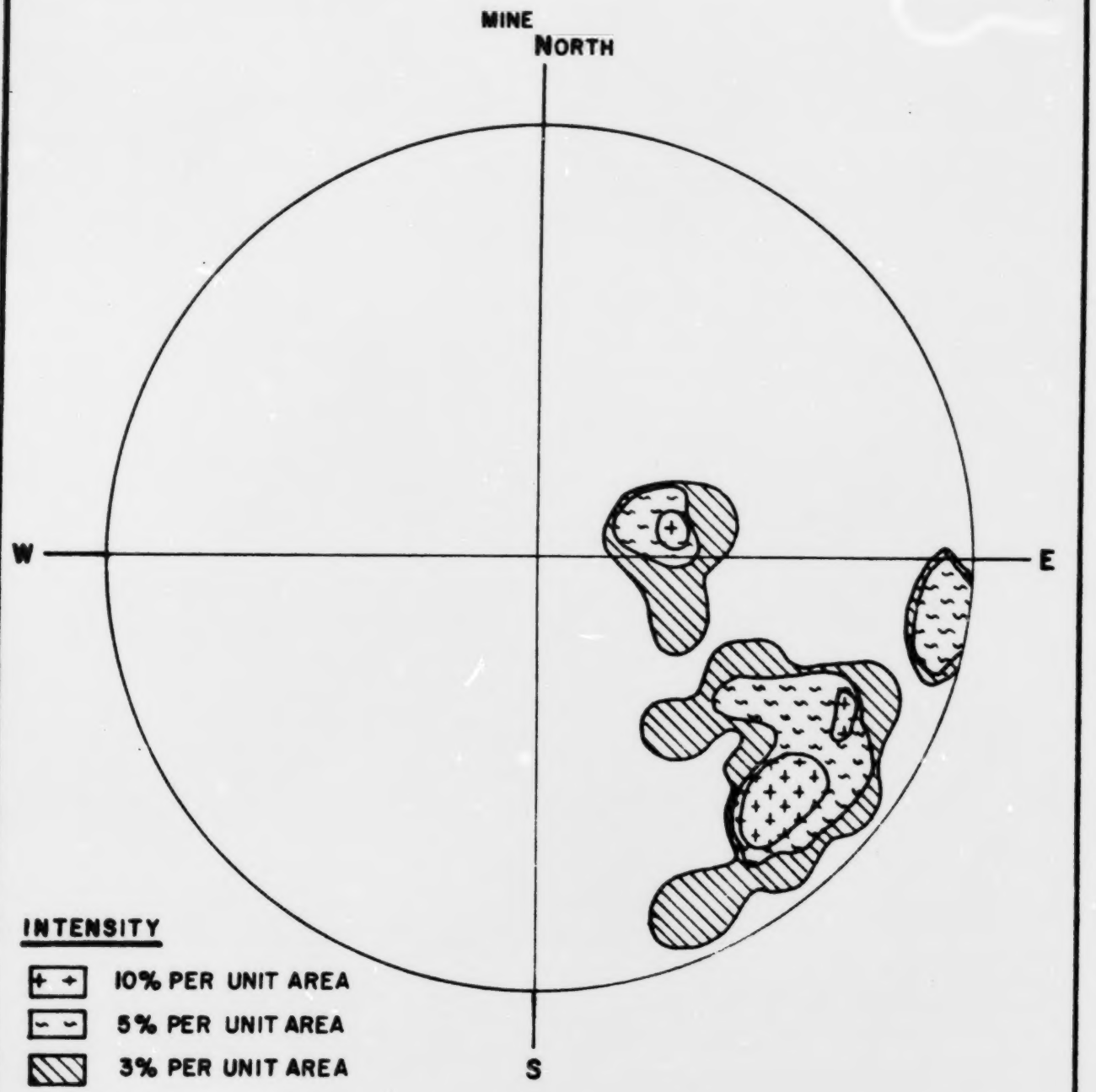

INTENSITY

- 10% PER UNIT AREA
- 5 % PER UNIT AREA
- 3% PER UNIT AREA

LOWER HEMISPHERE STEREOGRAPHIC PROJECTION

POLES TO JOINTING No. 3 ZONE DARK NORITE BRECCIA. 85 DATA POINTS

FIG: 12

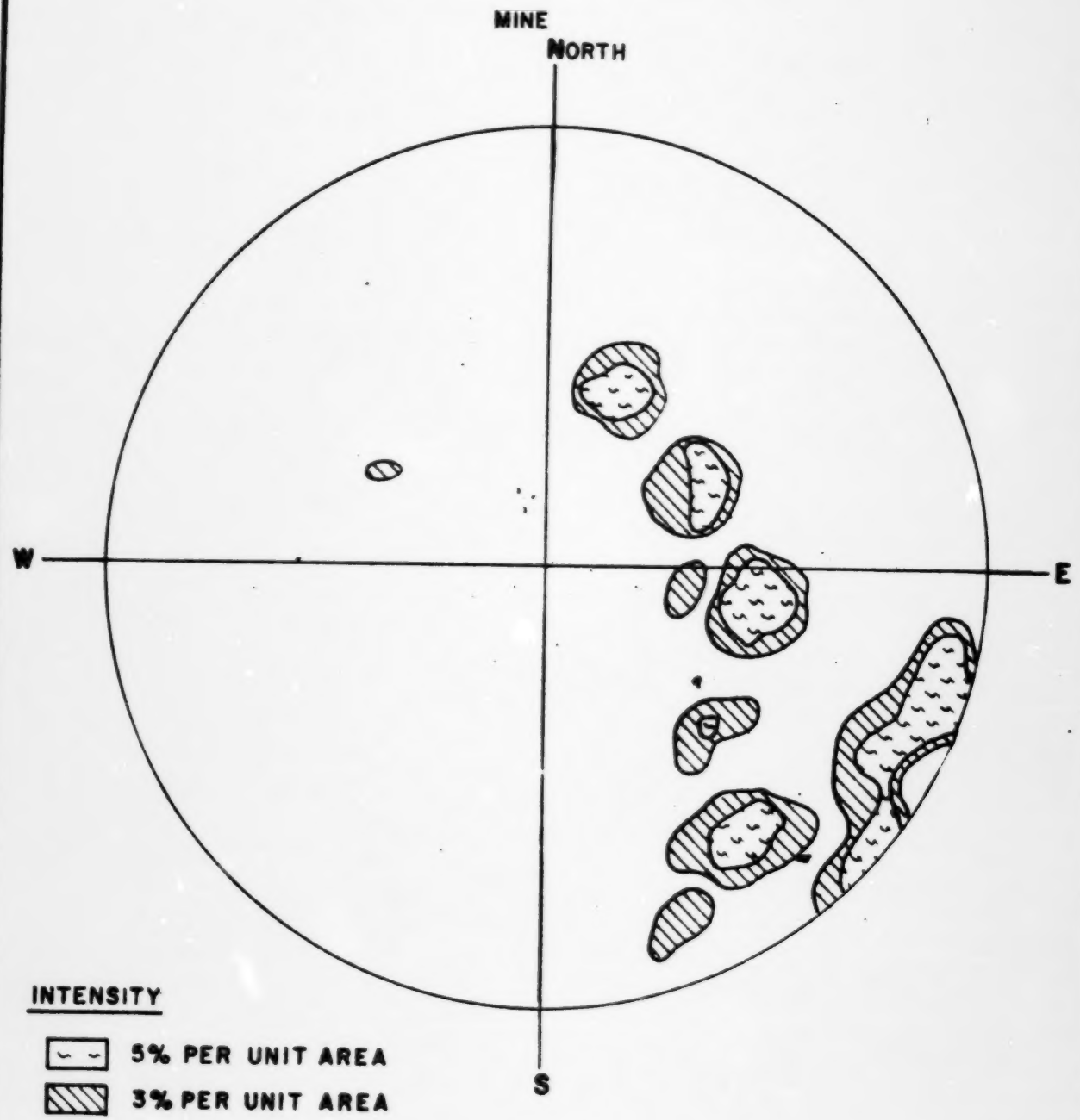



LOWER HEMISPHERE STEREOGRAPHIC PROJECTION

POLES TO JOINTING No. 3 ZONE DARK NORITE BRECCIAS

WITH DISSEMMINATED SULPHIDES. 41 DATA POINTS

FIG: 13

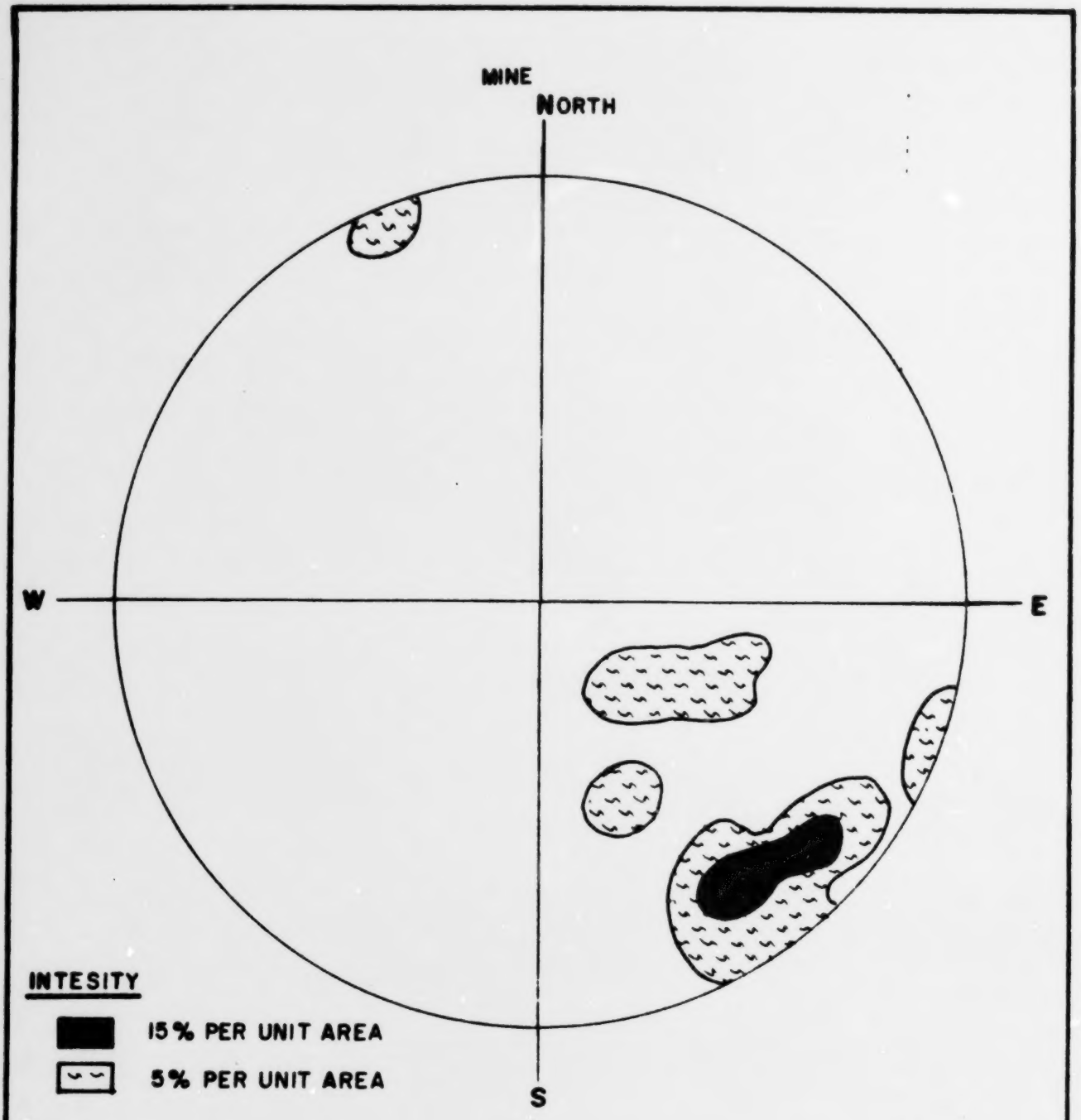



LOWER HEMISPHERE STEREOGRAPHIC PROJECTION

POLES TO JOINTING No. 3 ZONE DARK NORITE BRECCIAS

≥20% SULPHIDES. 44 DATA POINTS

FIG: 14

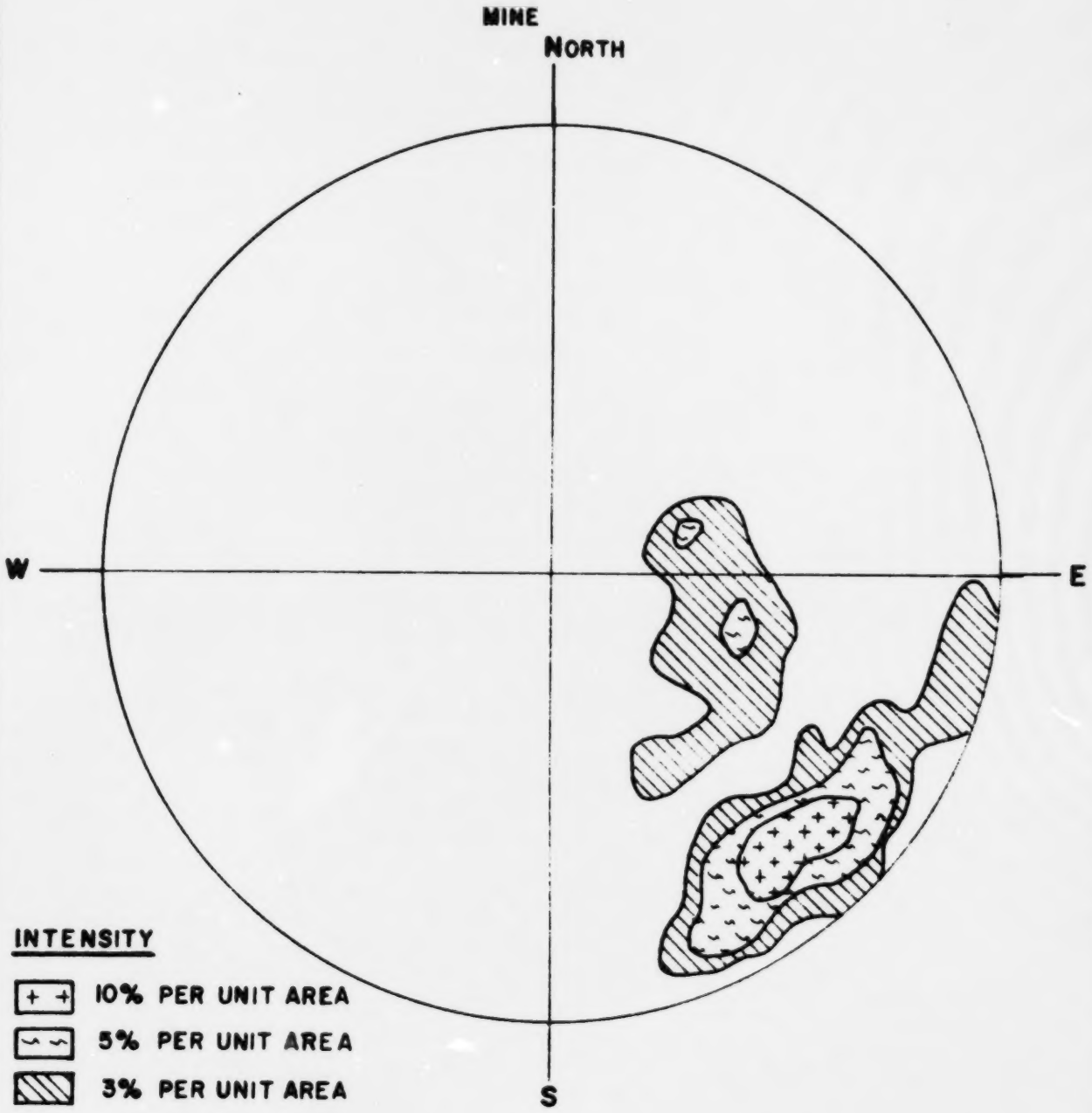



LOWER HEMISPHERE STEREOGRAPHIC PROJECTION

POLES TO JOINTING No. 3 ZONE LATE GRANITE BRECCIA

39 DATA POINTS

FIG: 15

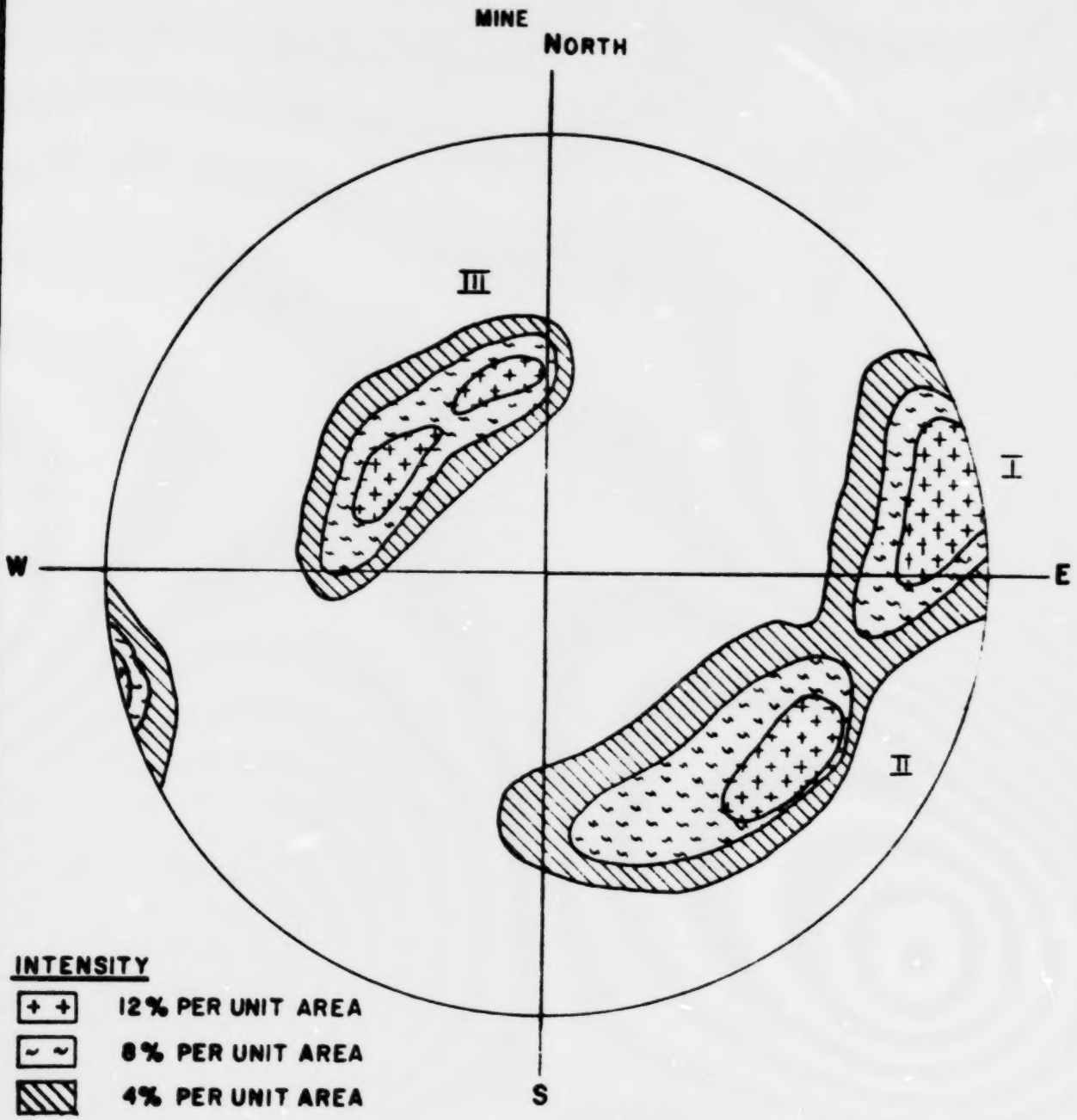



LOWER HEMISPHERE STEREOGRAPHIC PROJECTION

POLES TO JOINTING No.3 ZONE DARK NORITE BRECCIA

AND LATE GRANITE BRECCIA. 124 DATA POINTS

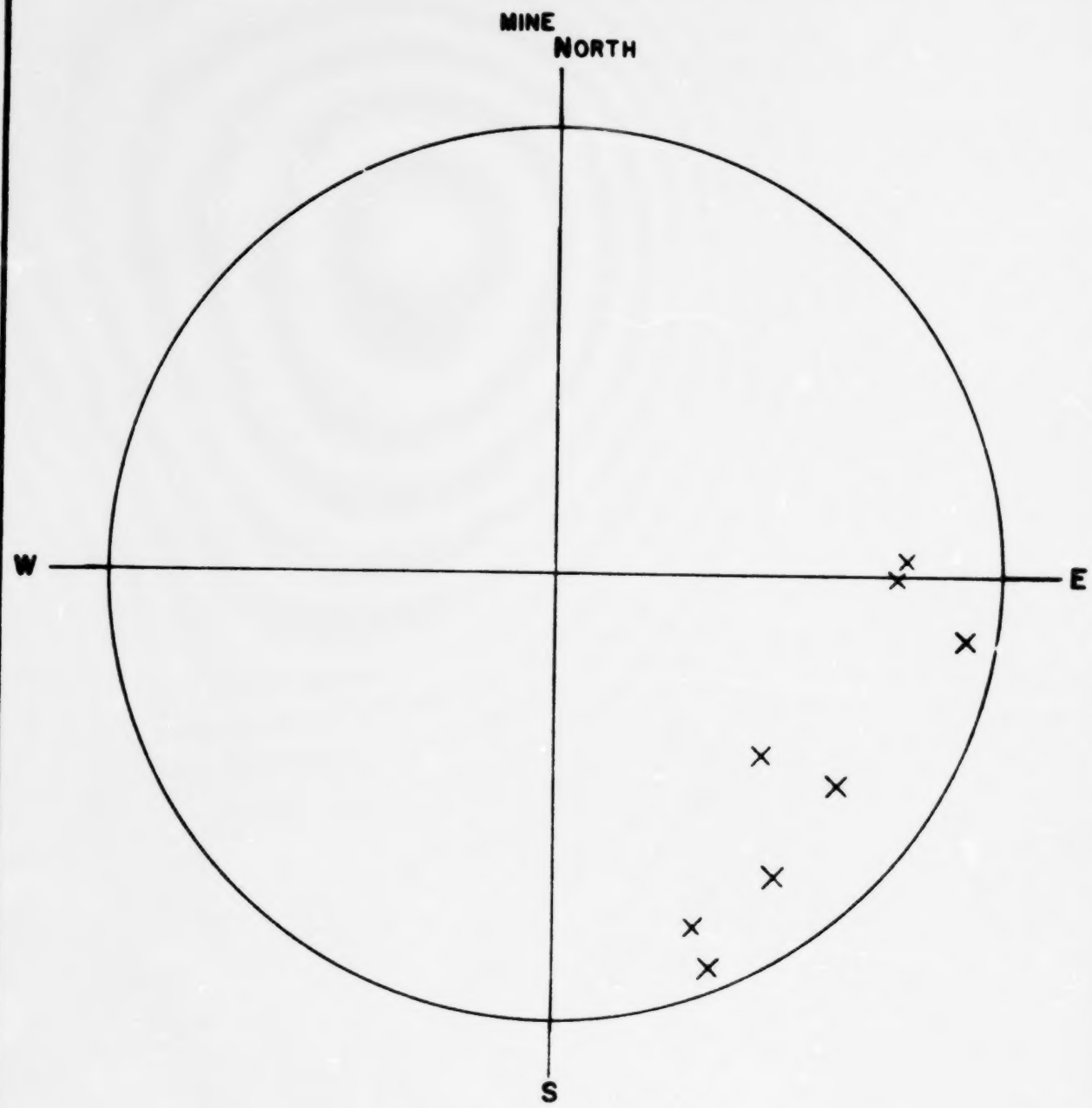
FIG: 16




LOWER HEMISPHERE STEREOGRAPHIC PROJECTION

POLES TO NORITE JOINTING, 2050 AND 3050 EAST EXPLORATION DRIFTS. 377 DATA POINTS

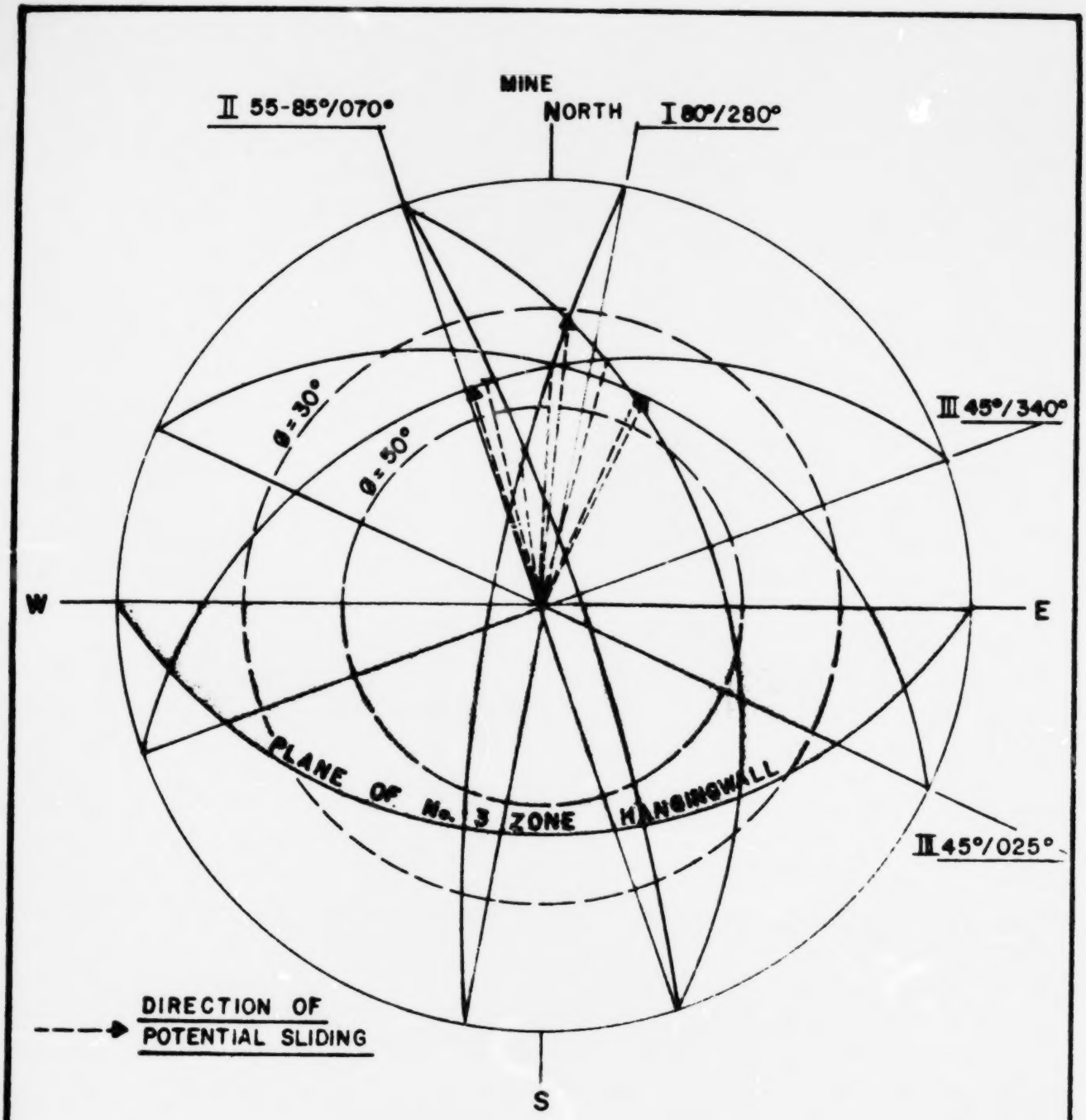
FIG: 17




LOWER HEMISPHERE STEREOGRAPHIC PROJECTION

POLES TO FAULTING, No. 3 ZONE 8 DATA POINTS

FIG: 18




LOWER HEMISPHERE PROJECTION OF MAJOR JOINT SETS — ONAPING No. 3 ZONE NORITES

FIG : 19

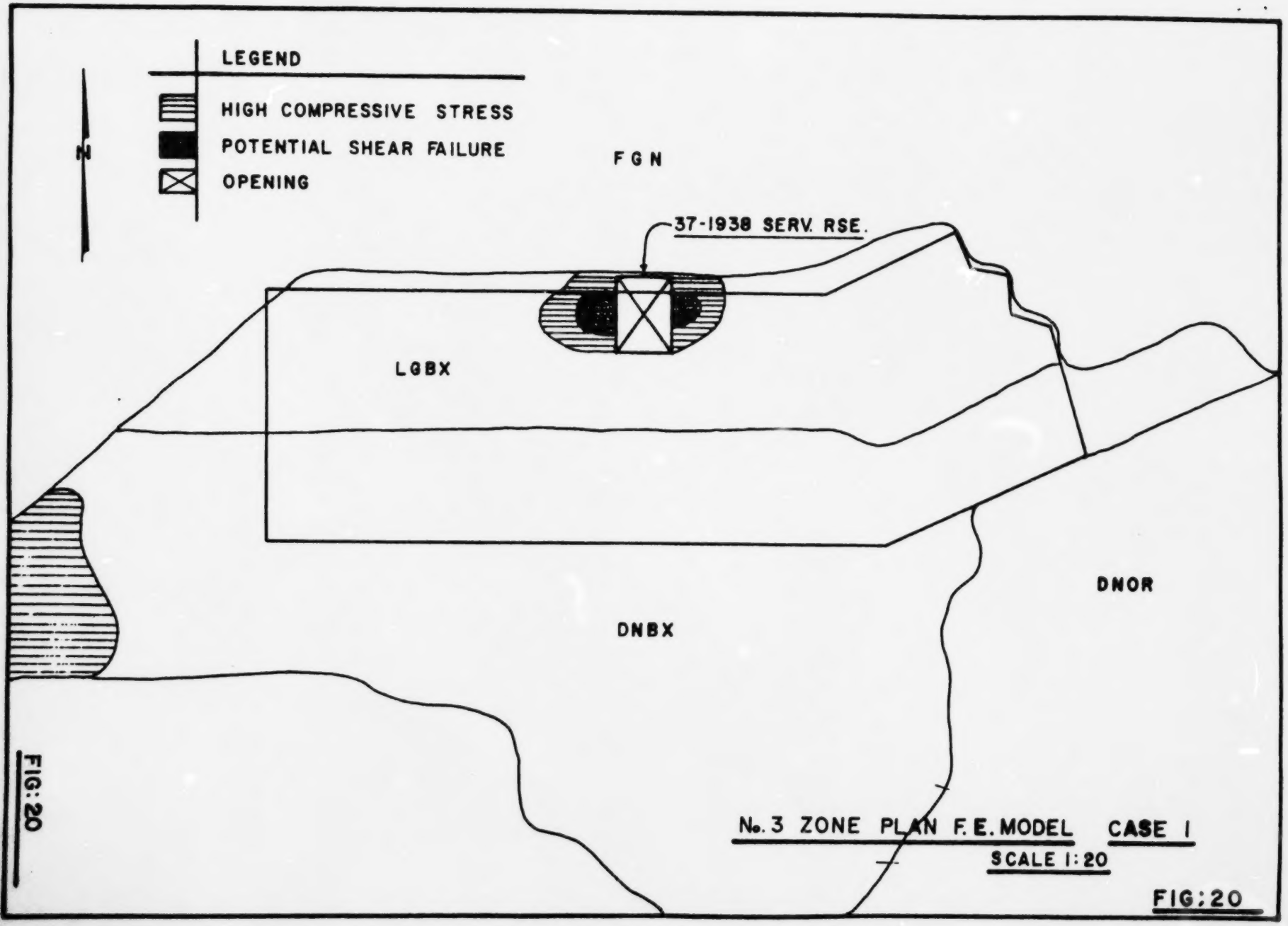



FIG:20

No. 3 ZONE PLAN F.E. MODEL CASE 1

SCALE 1:20

FIG:20

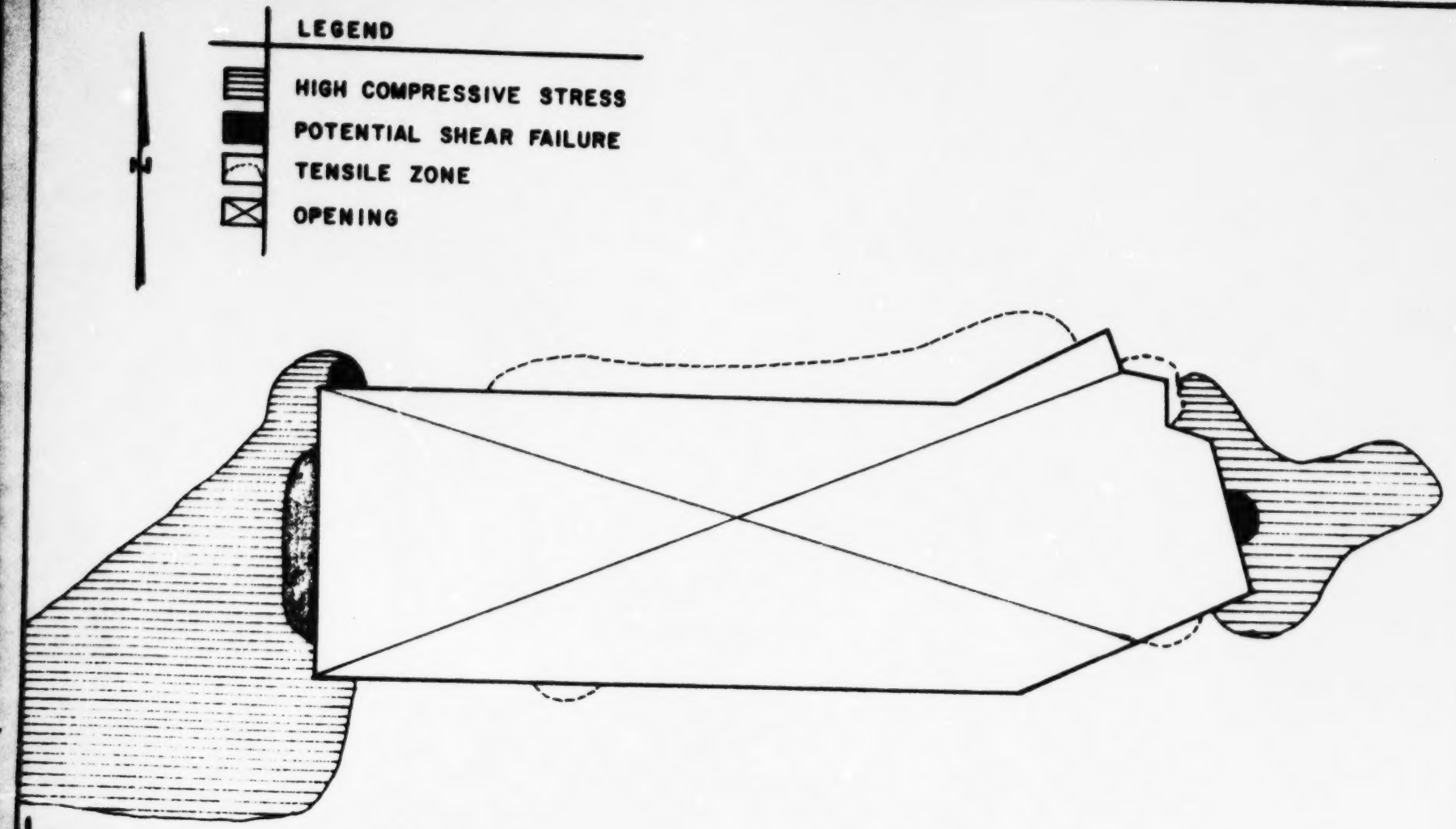
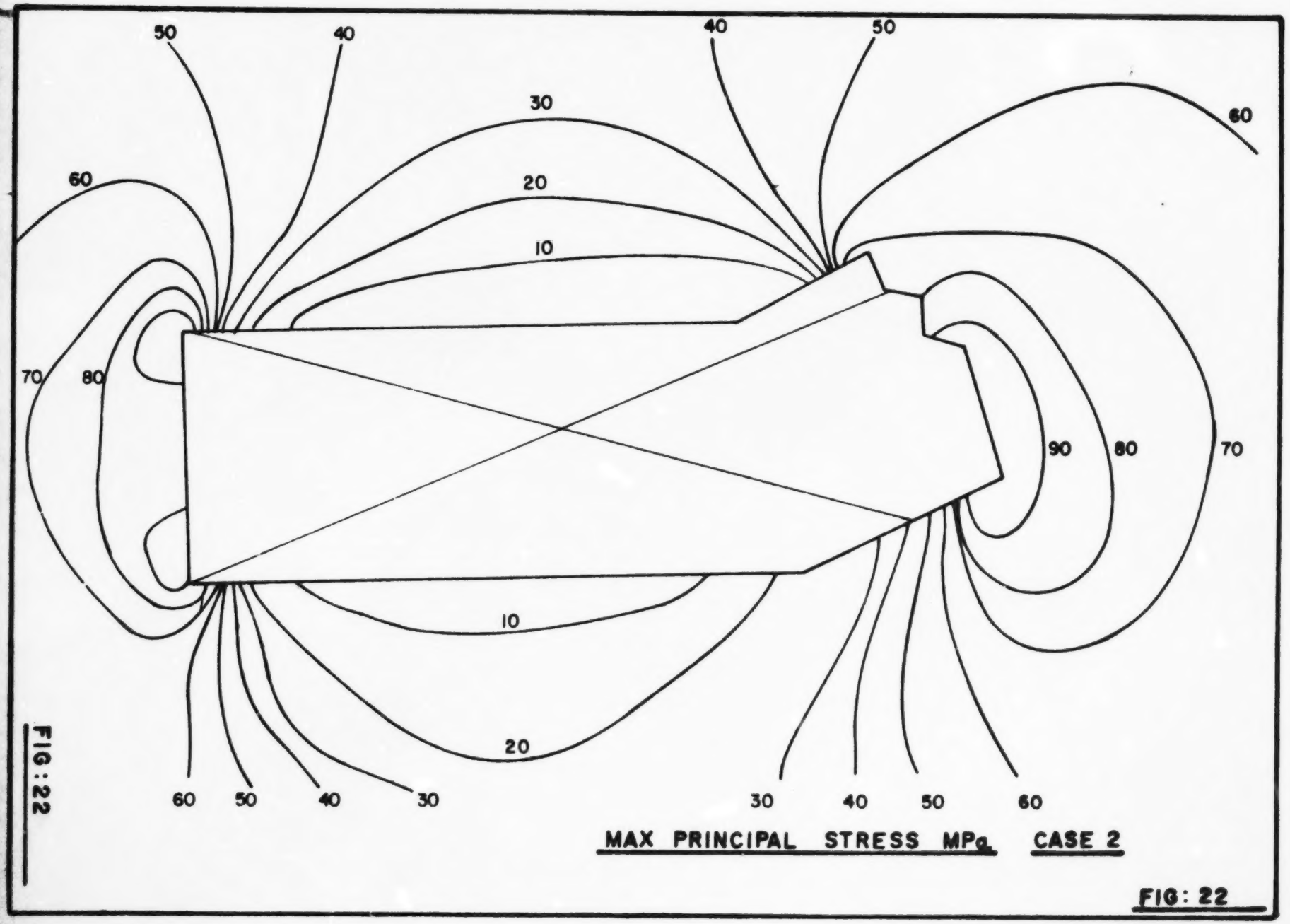



FIG: 21

No.3 ZONE PLAN F.E. MODEL CASE 2

SCALE 1:20

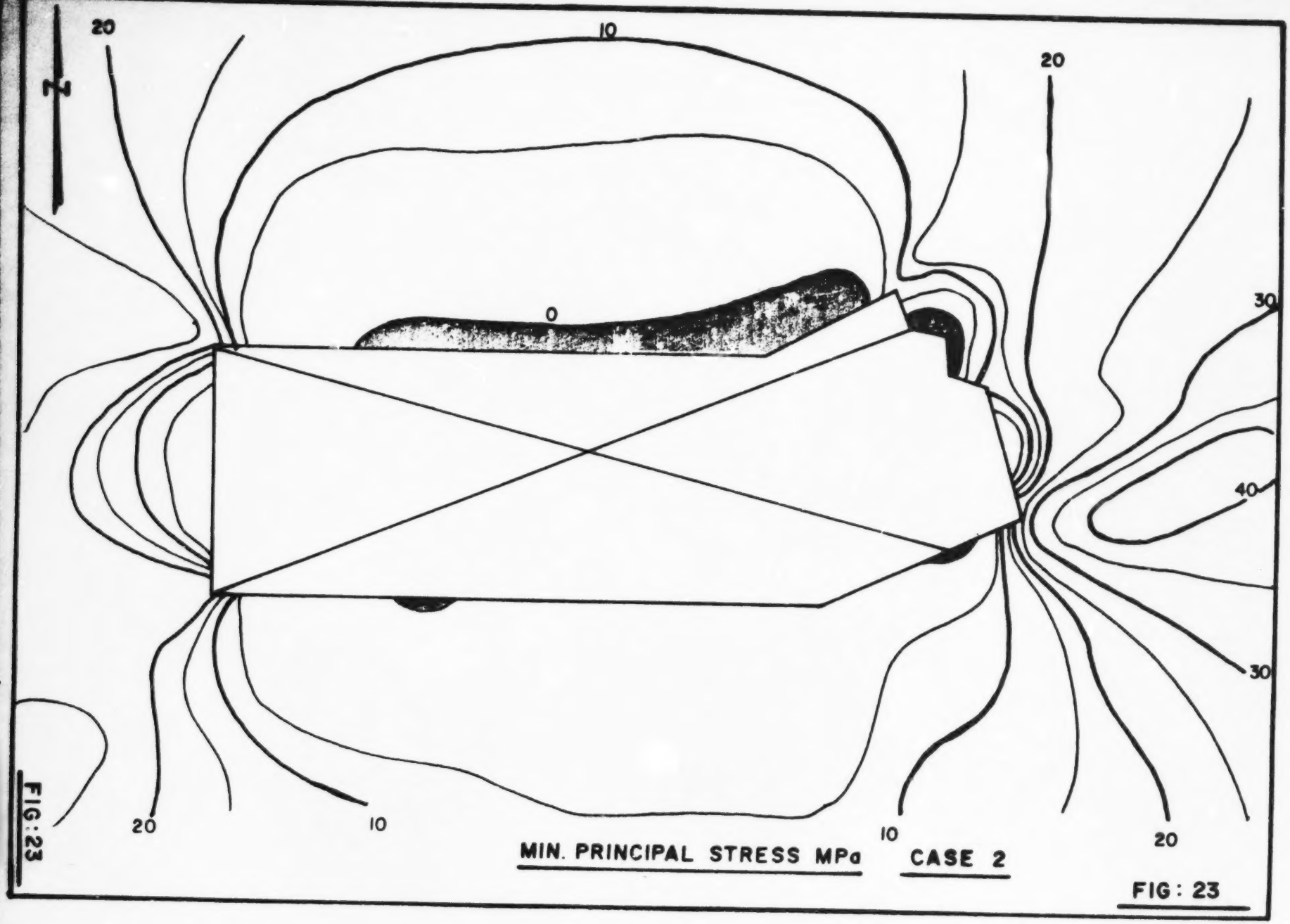
FIG: 21




MAX PRINCIPAL STRESS MPa CASE 2

FIG: 22

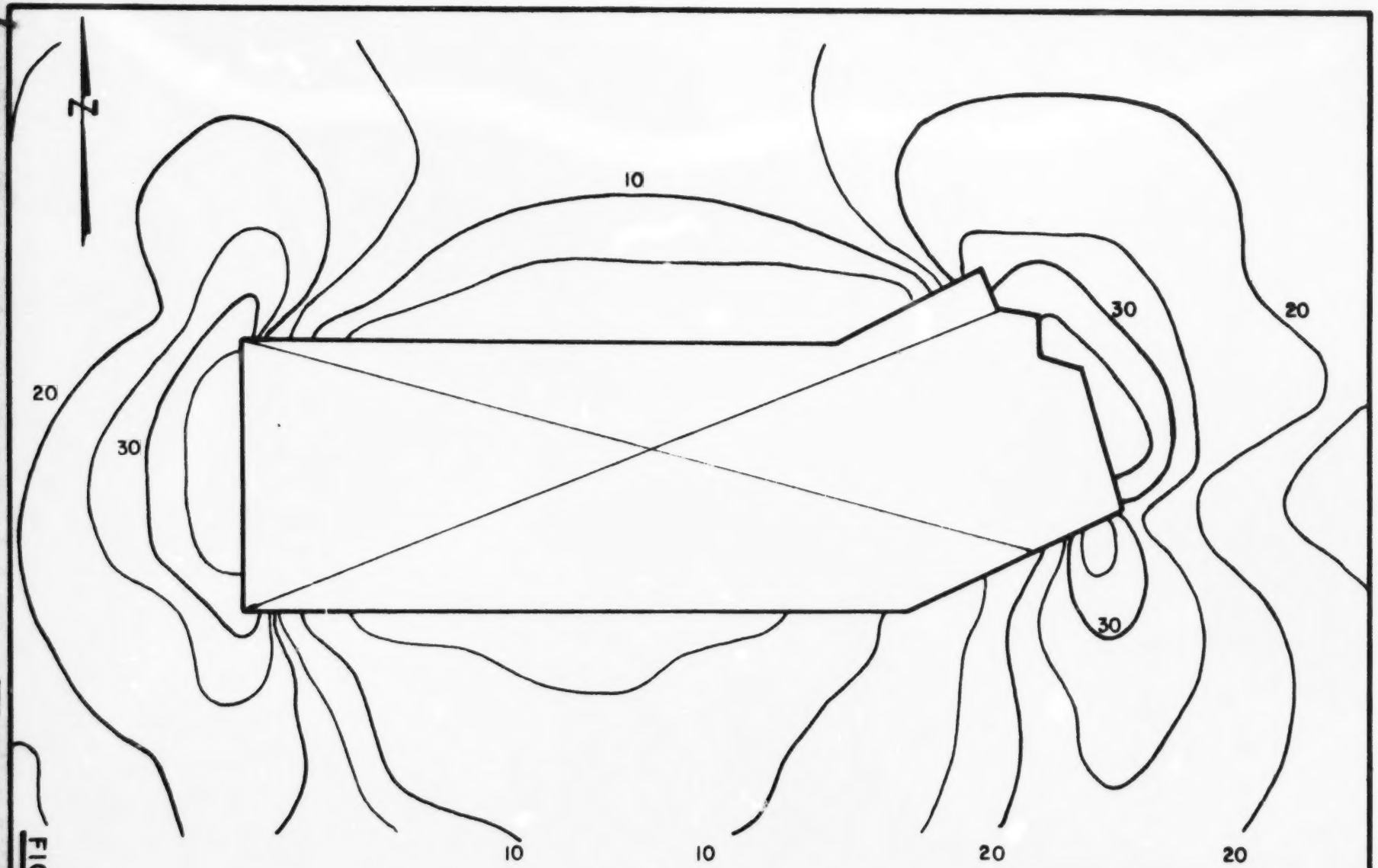
FIG: 22




MIN. PRINCIPAL STRESS MPa CASE 2

FIG: 23

FIG: 23




MAX. SHEAR MPa CASE 2

FIG: 24

FIG:24

LEGEND

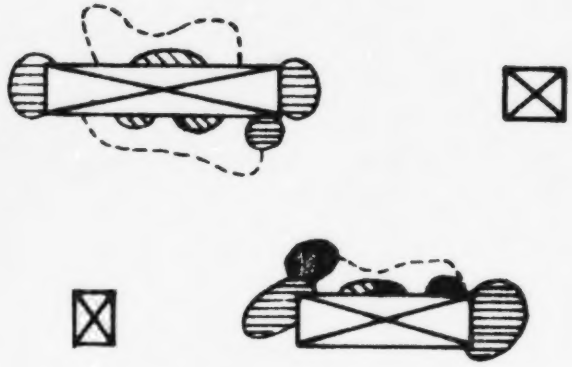
POTENTIAL TENSILE FAILURE

HIGH COMPRESSIVE STRESS

POTENTIAL SHEAR FAILURE

TENSILE ZONE

OPENING



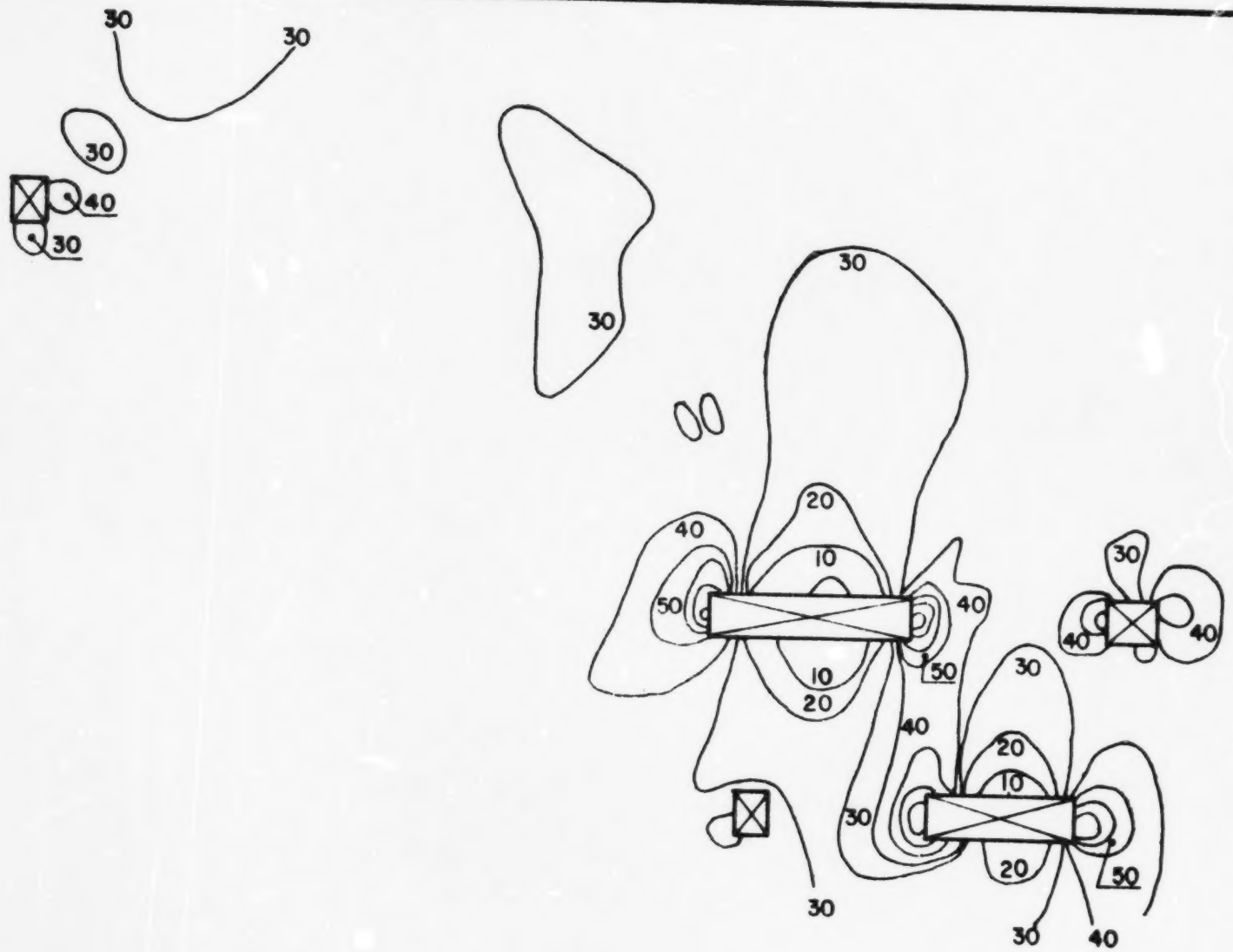
N.3 ZONE TRANSVERSE F.E. MODEL

CASE I DEVELOPMENT OPEN K=1.5

SCALE 1=40

FIG: 25

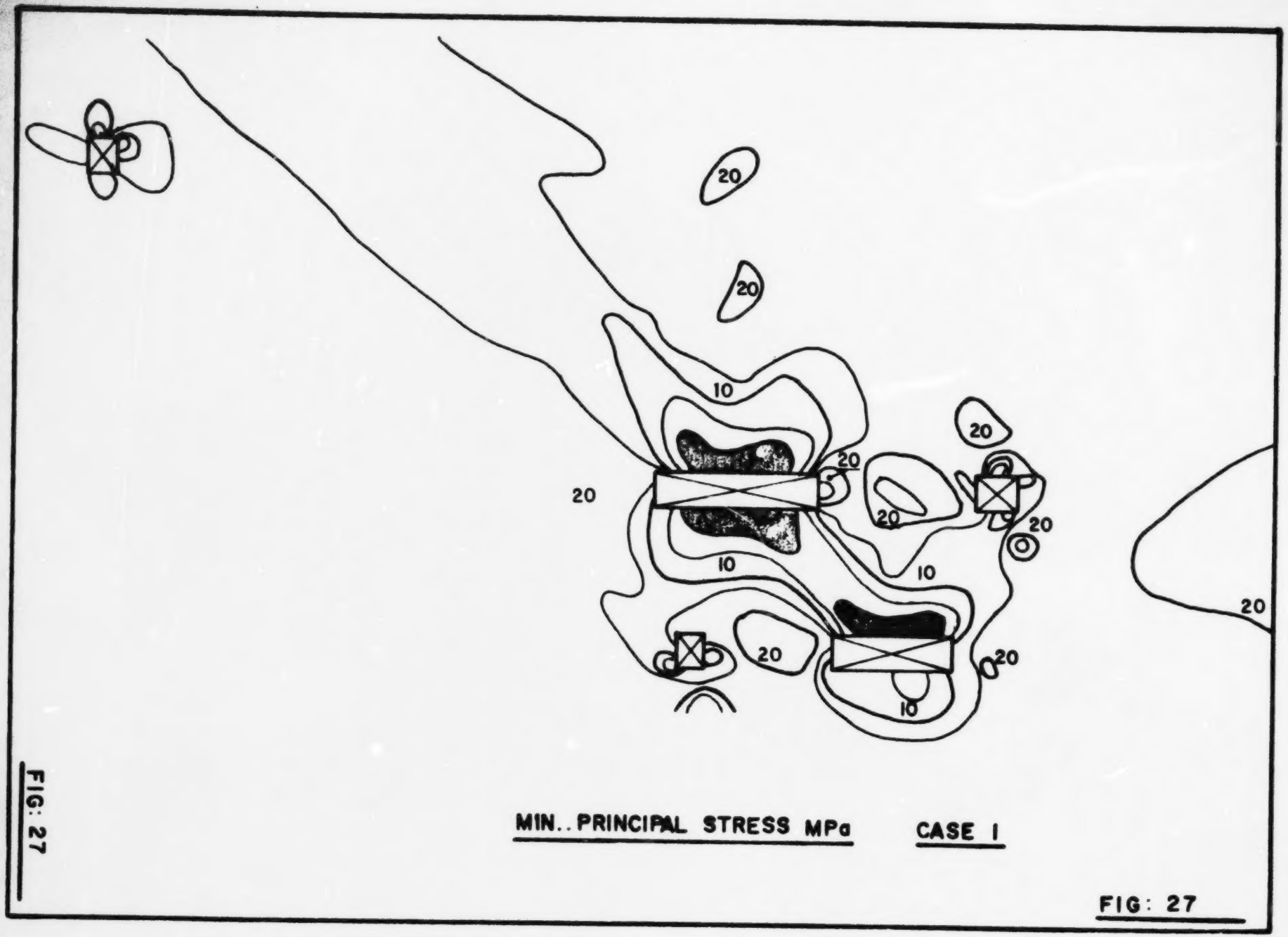
FIG: 25




MAX. PRINCIPAL STRESS MPa CASE I

FIG: 26

FIG: 26




MIN.. PRINCIPAL STRESS MPa CASE I

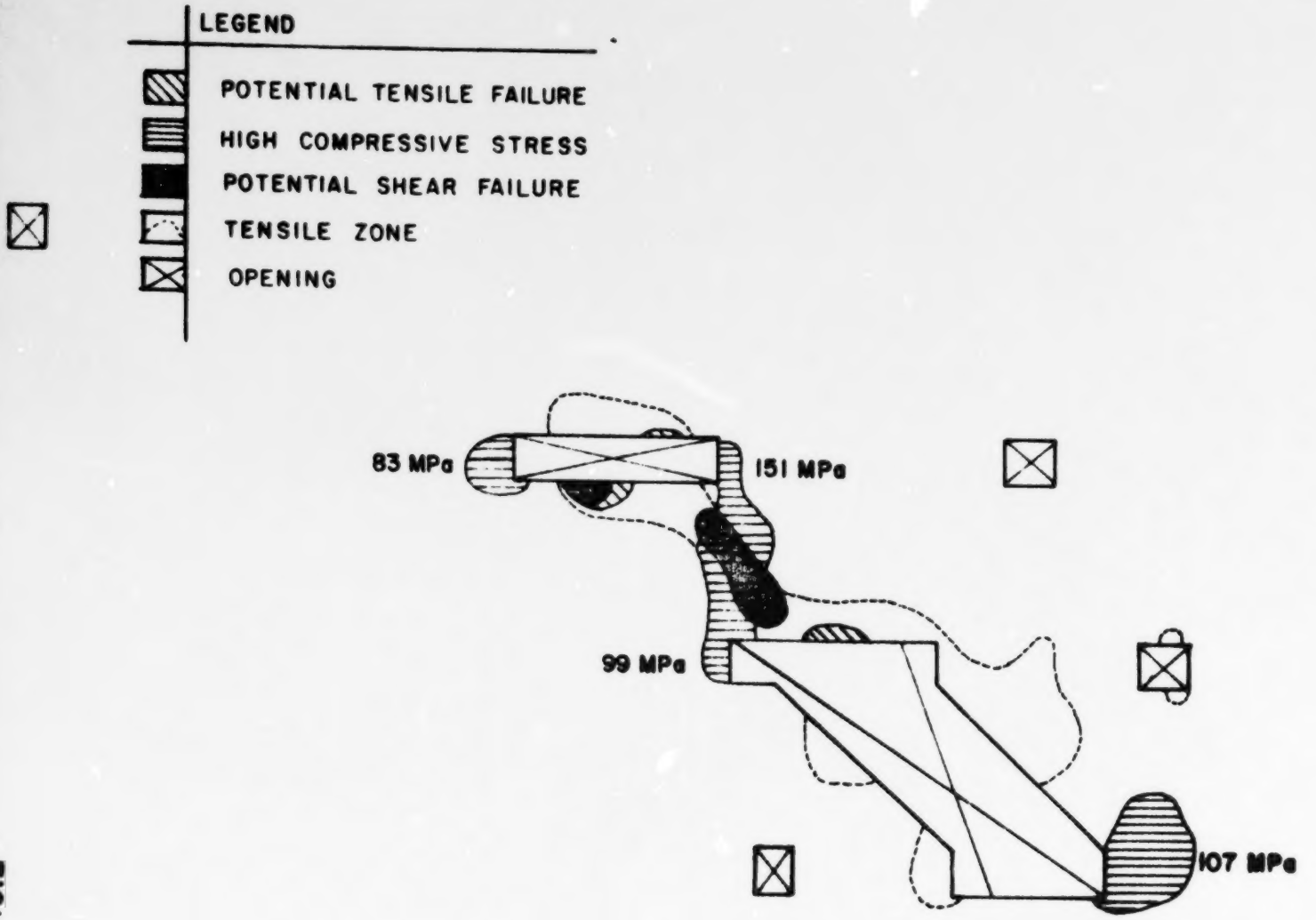
FIG: 27

FIG: 27

MAX SHEAR MPa CASE 1

FIG: 28

FIG:28



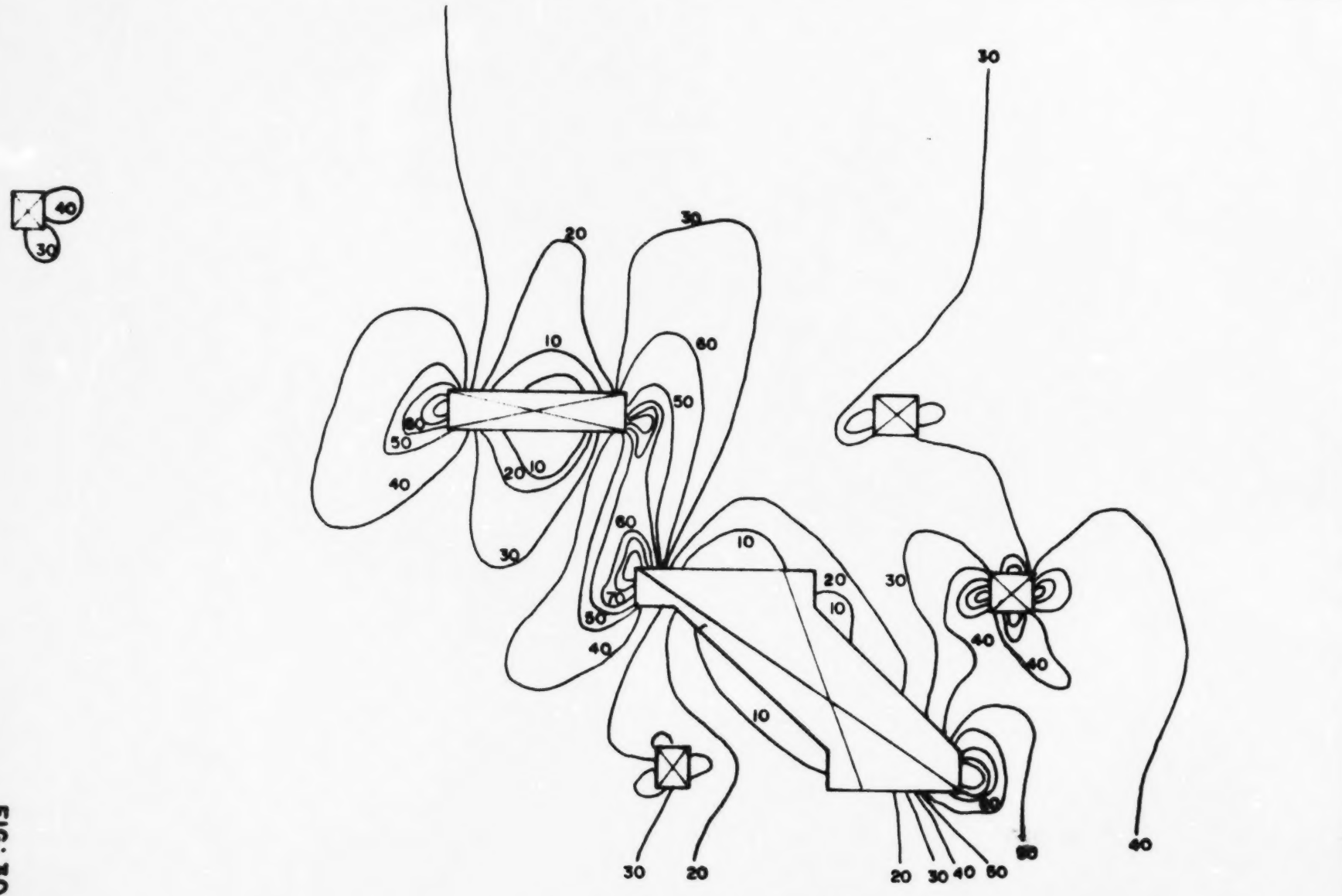

No. 3 ZONE TRANSVERSE F. E. MODEL

CASE 2 - 1ST PANEL OPEN K = 1.5

SCALE 1 = 40

FIG:29

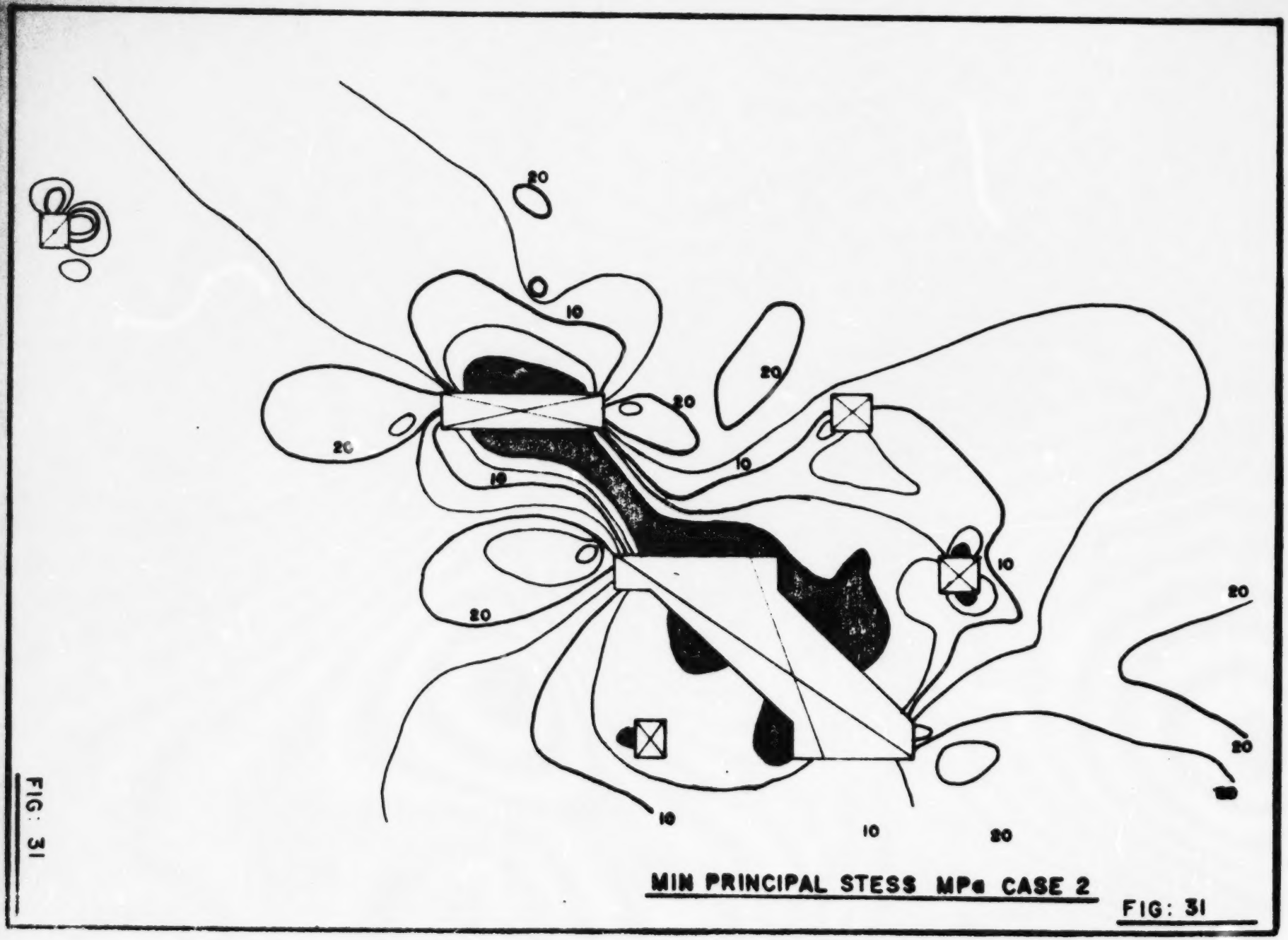
FIG: 29




MAX PRINCIPAL STRESS MPa CASE 2

FIG: 30

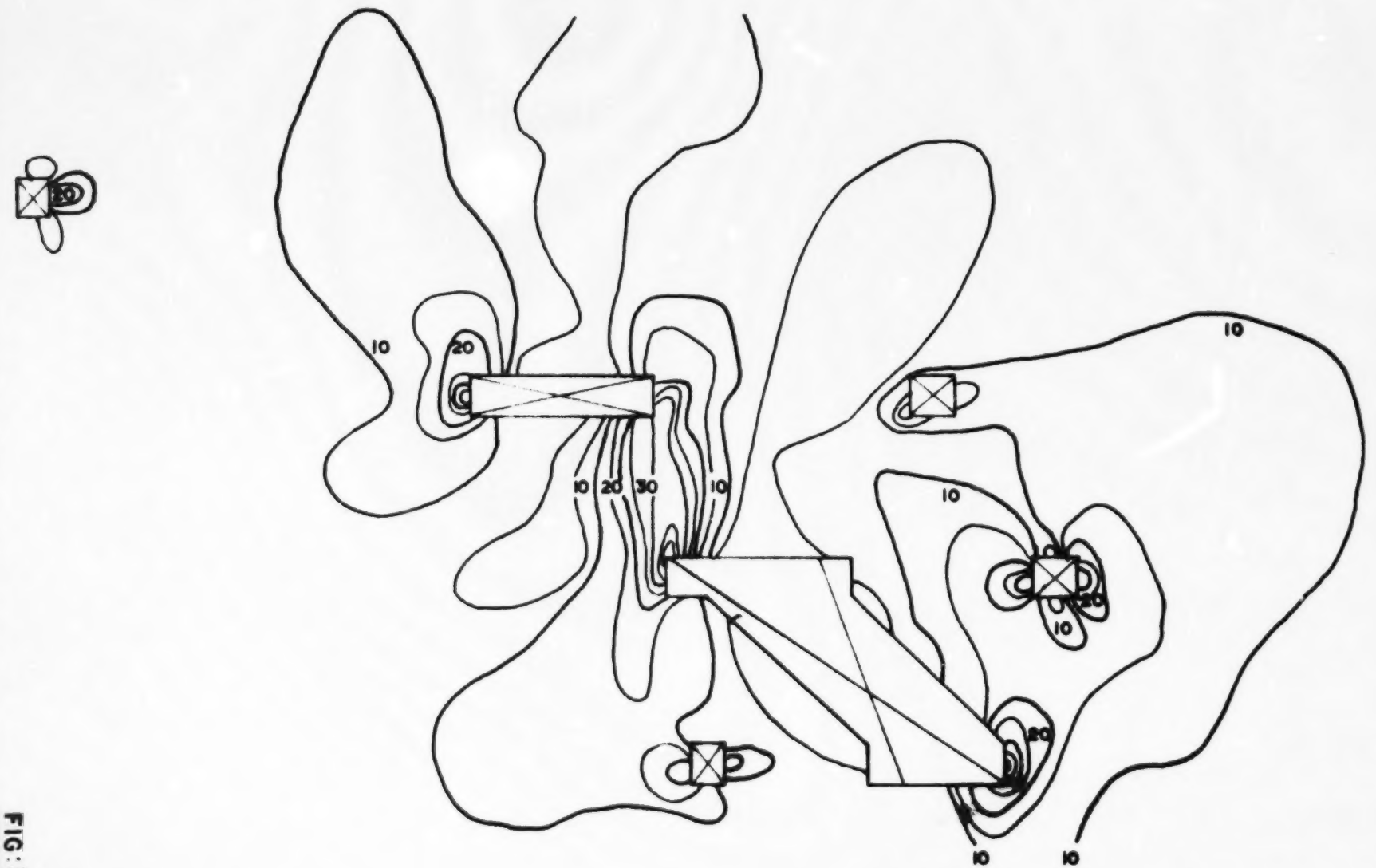
FIG: 30




MIN PRINCIPAL STESS MPa CASE 2

FIG: 31

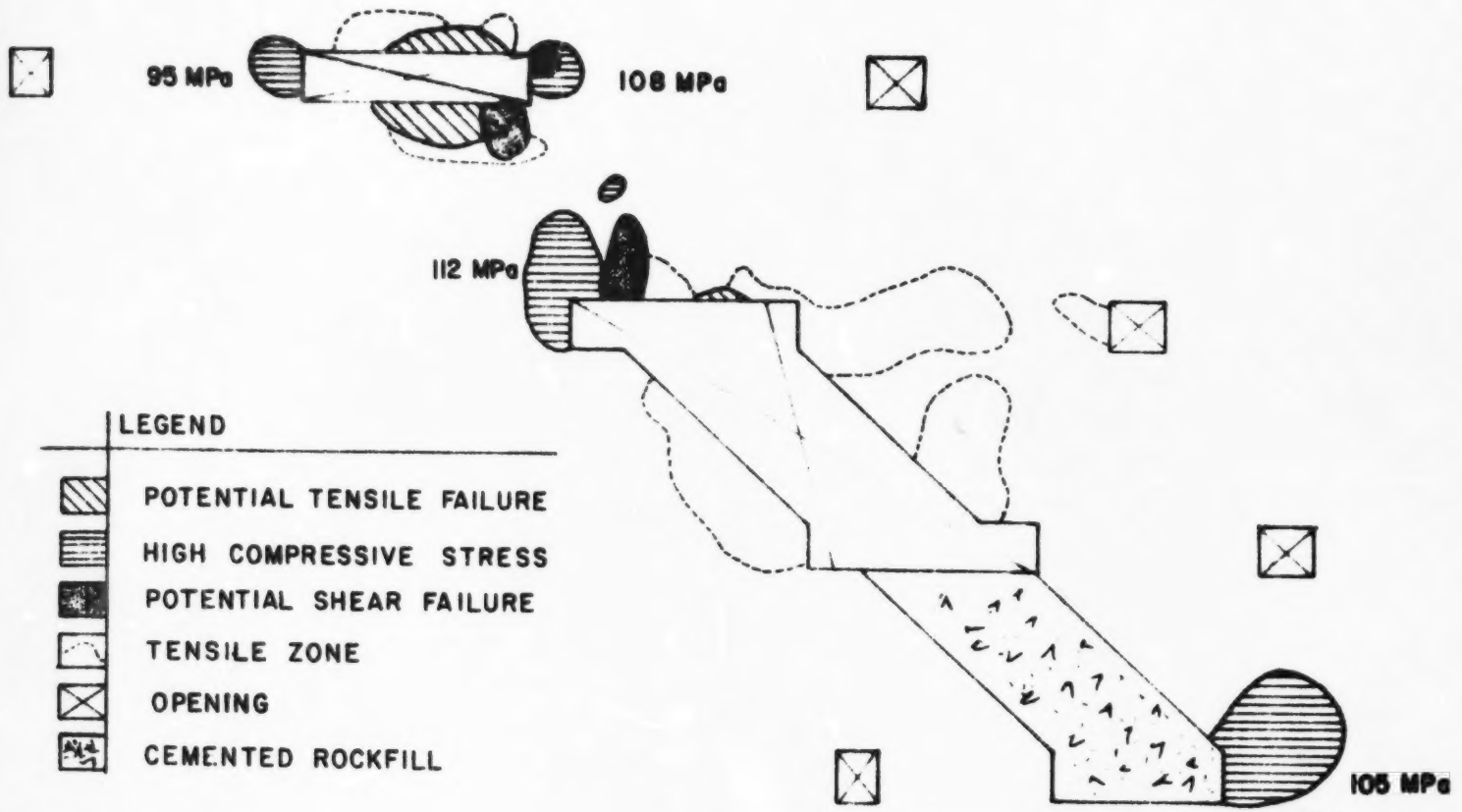
FIG: 31




MAX SHEAR MPa CASE B

FIG: 32

FIG: 32



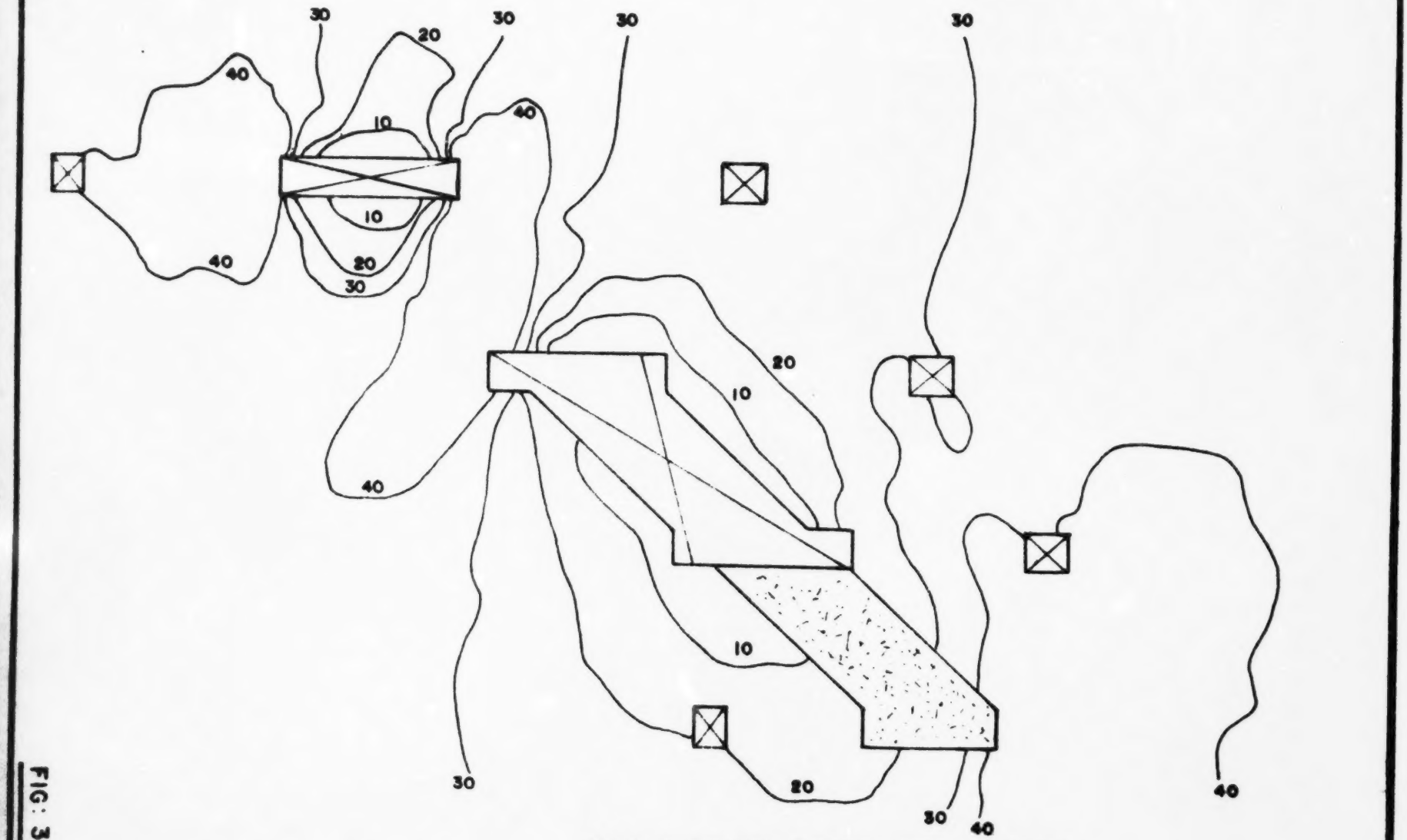

No. 3 ZONE TRANSVERSE F.E. MODEL

CASE 3 - 1st PANEL FILLED, 2nd PANEL OPEN K-1.5

SCALE 1=40

FIG: 33

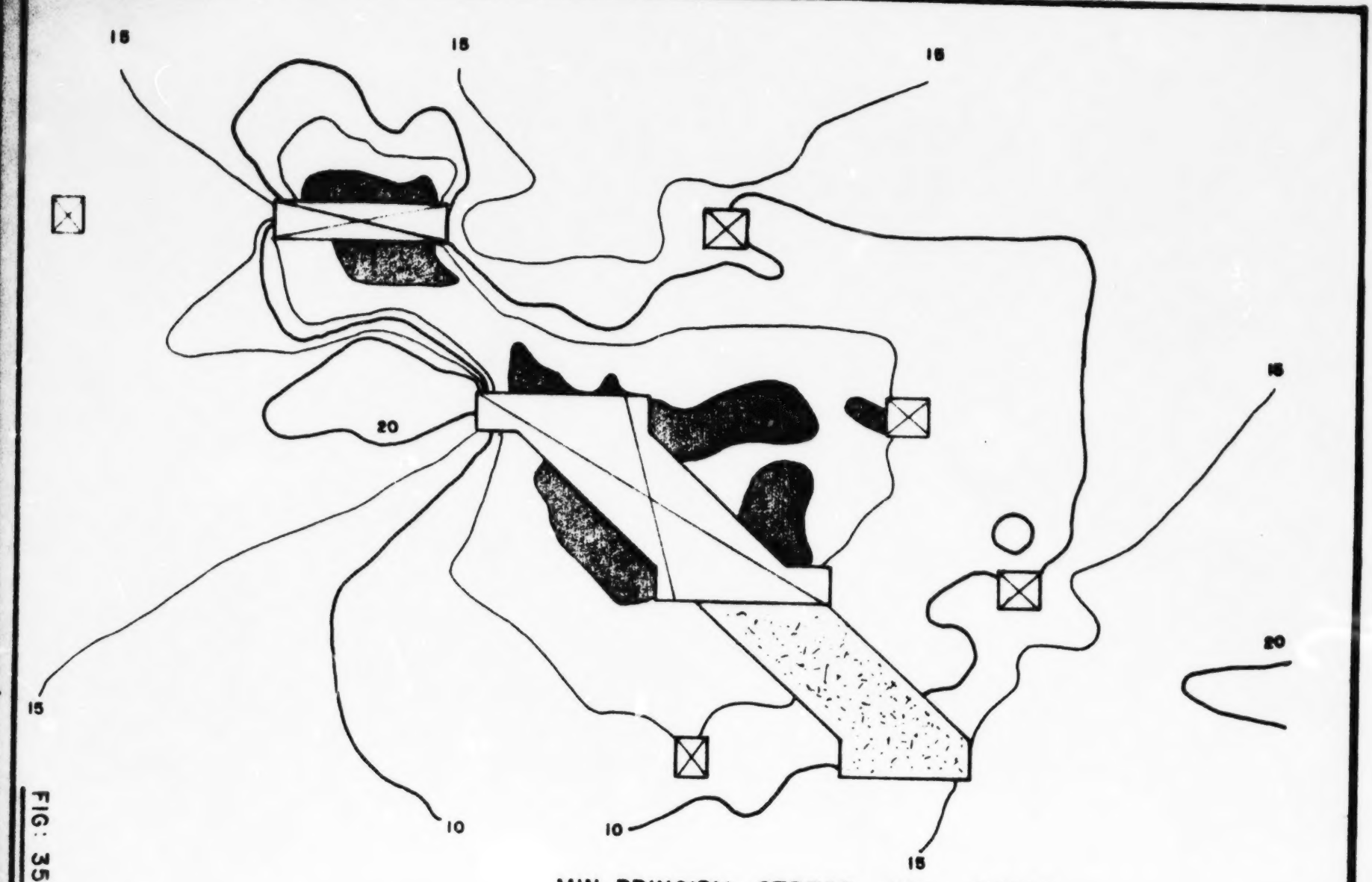
FIG 33




MAX. PRINCIPAL STRESS MPa CASE 3

FIG: 34

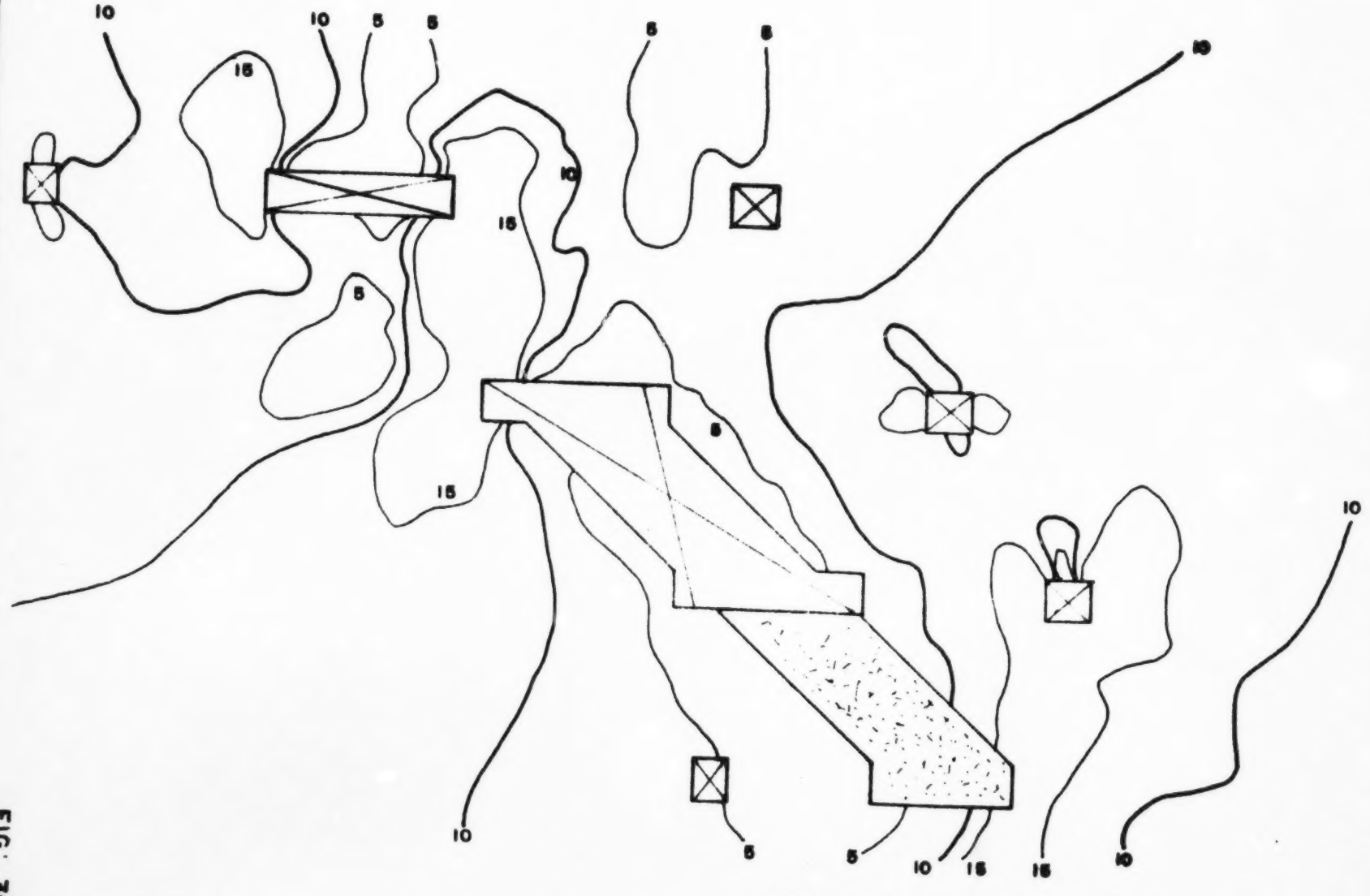
FIG: 34




MIN PRINCIPAL STRESS MPa CASE 3

FIG: 35

FIG: 35




MAX SHEAR MPa CASE 3

FIG: 36

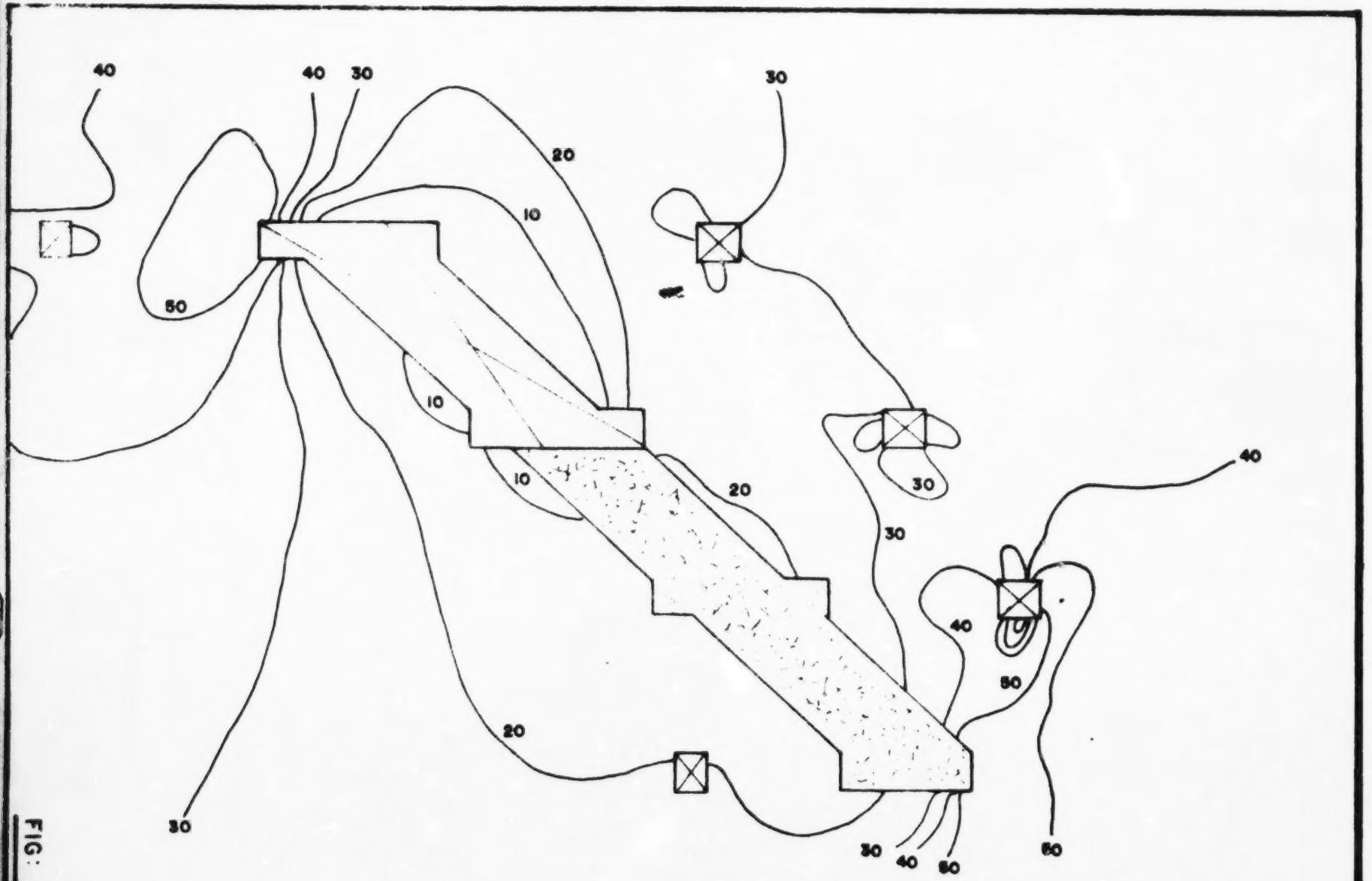
FIG: 36

No. 3 ZONE TRANSVERSE F.E. MODEL

CASE 4 2ND PANEL FILLED, 3RD PANEL OPEN K=1.5

FIG: 37

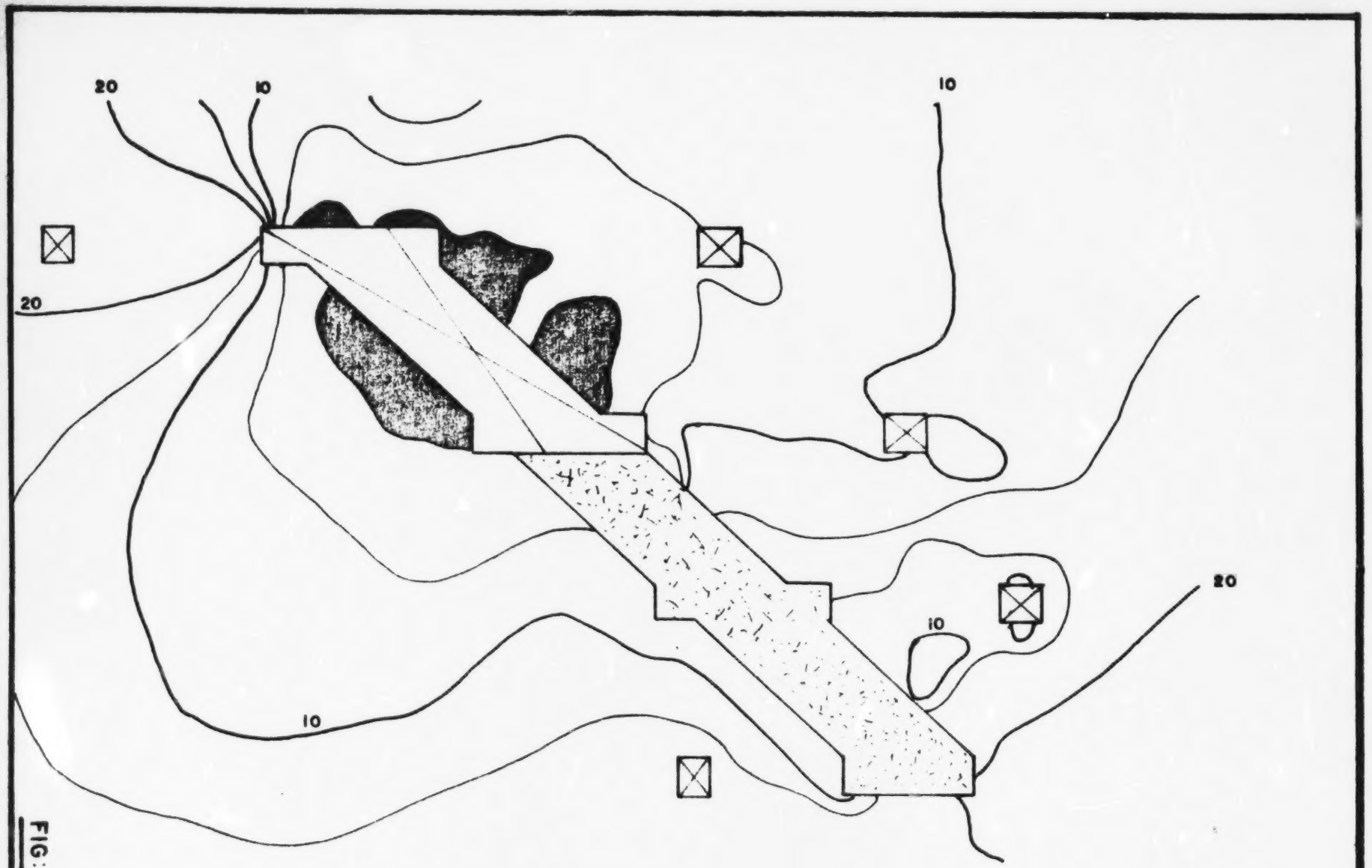
FIG: 37




MAX PRINCIPAL STRESS MPa CASE 4

FIG: 38

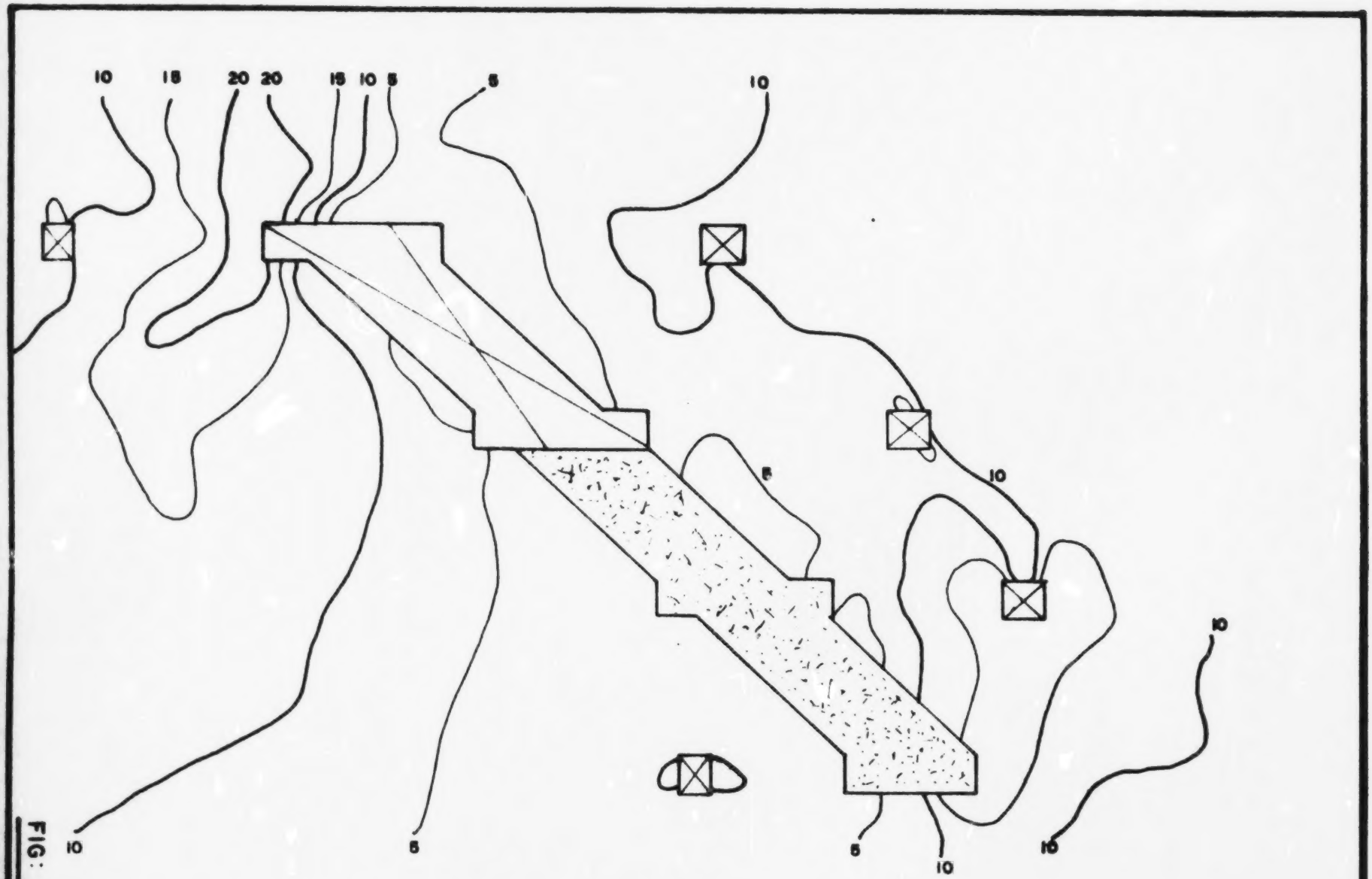
FIG: 38




MIN PRINCIPAL STESS MPa CASE 4

FIG: 39

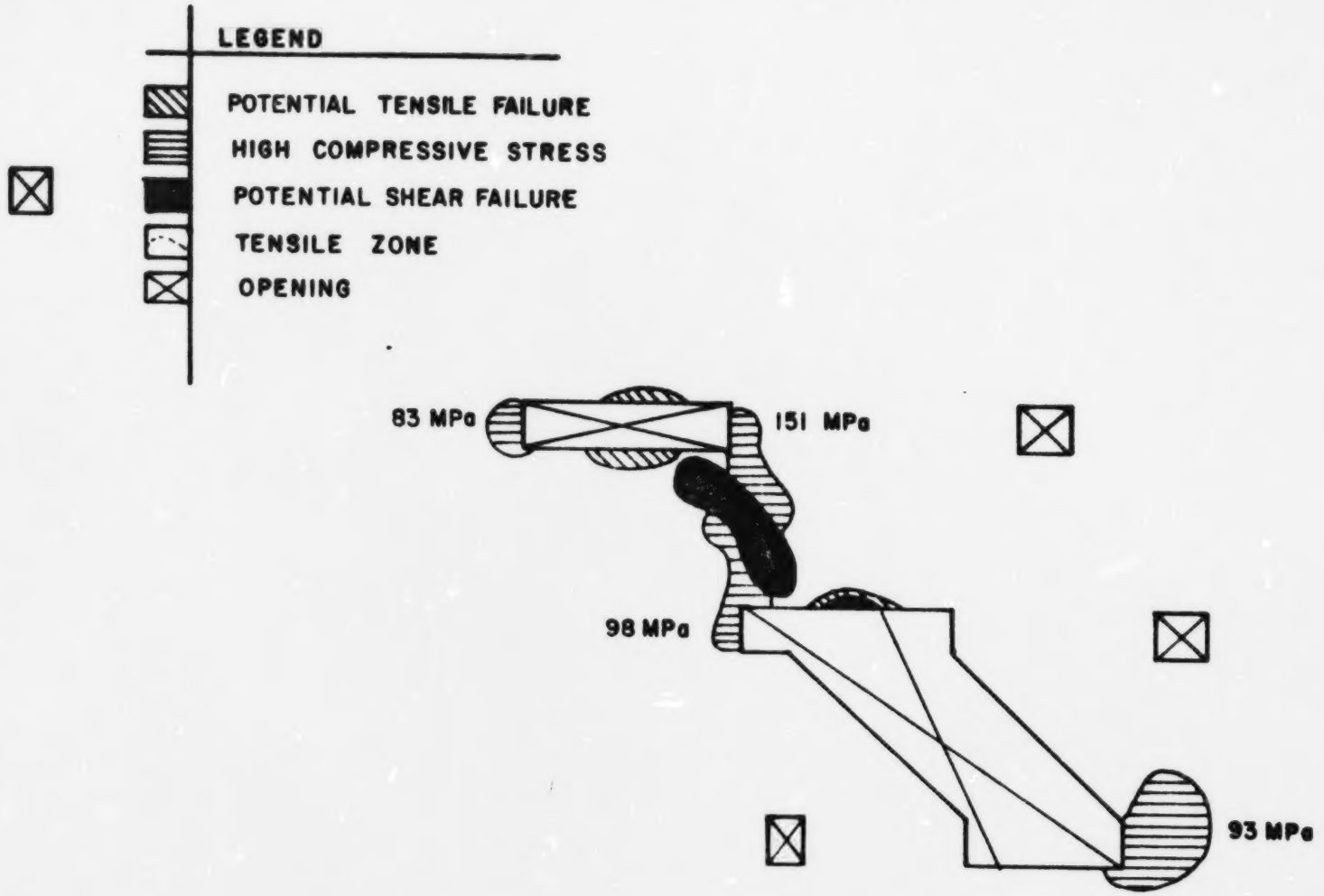
FIG: 39




MAX SHEAR MPa CASE 4

FIG: 40

FIG: 40




No. 3 ZONE TRANSVERSE F.E. MODEL

CASE 5 - 1ST PANEL OPEN K=1.2

SCALE 1=40

FIG: 41

FIG: 41

LEGEND

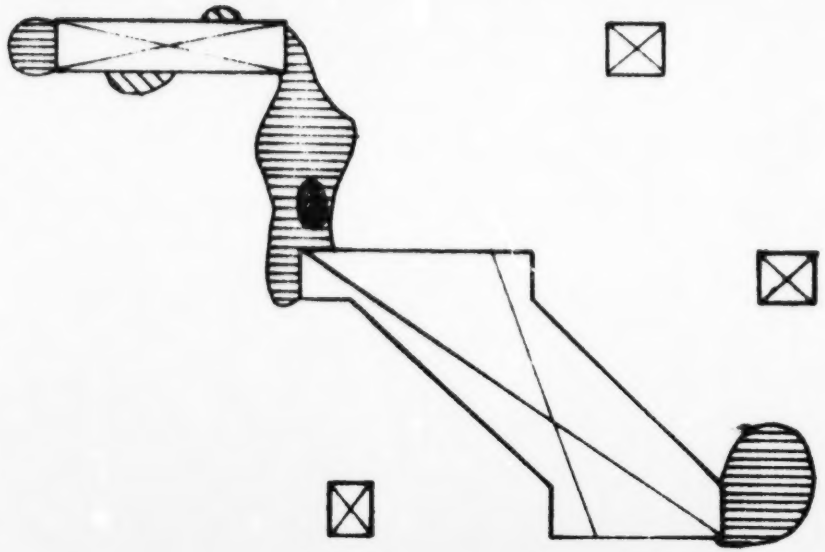
POTENTIAL TENSILE FAILURE

HIGH COMPRESSIVE STRESS

POTENTIAL SHEAR FAILURE

TENSILE ZONE

OPENING



No.3 ZONE TRANSVERSE F.E. MODEL

CASE 6 – 1st. PANEL OPEN K = 2.5

SCALE 1=40

FIG:42

FIG:42

LEGEND

POTENTIAL TENSILE FAILURE

HIGH COMPRESSIVE STRESS

POTENTIAL SHEAR FAILURE

TENSILE ZONE

OPENING

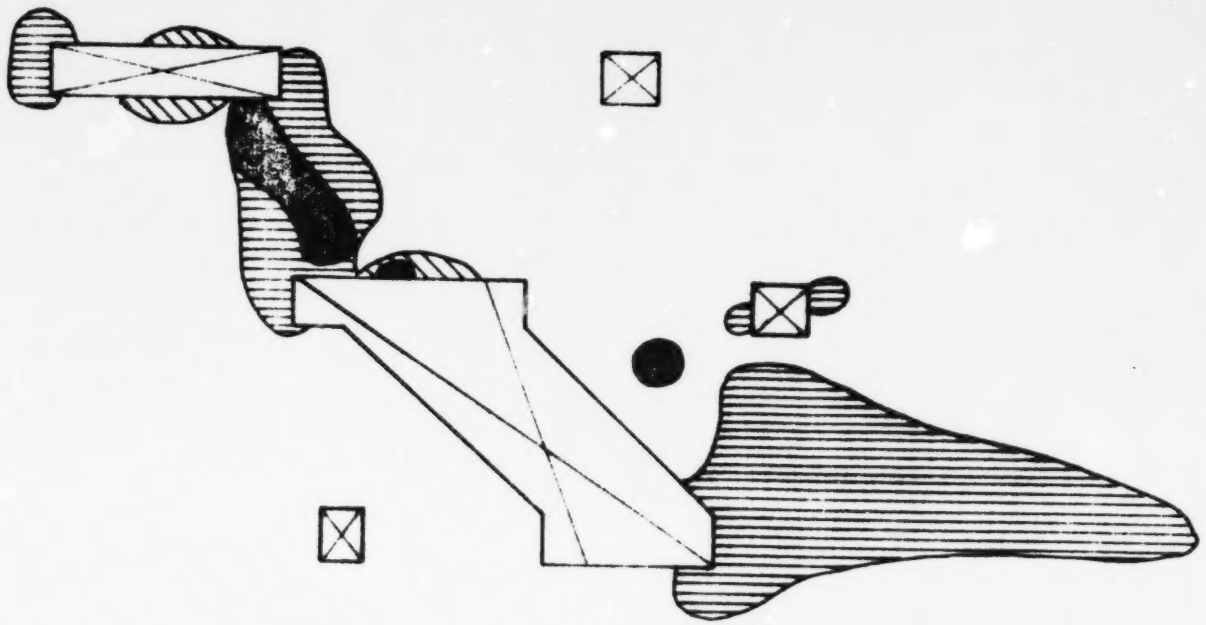


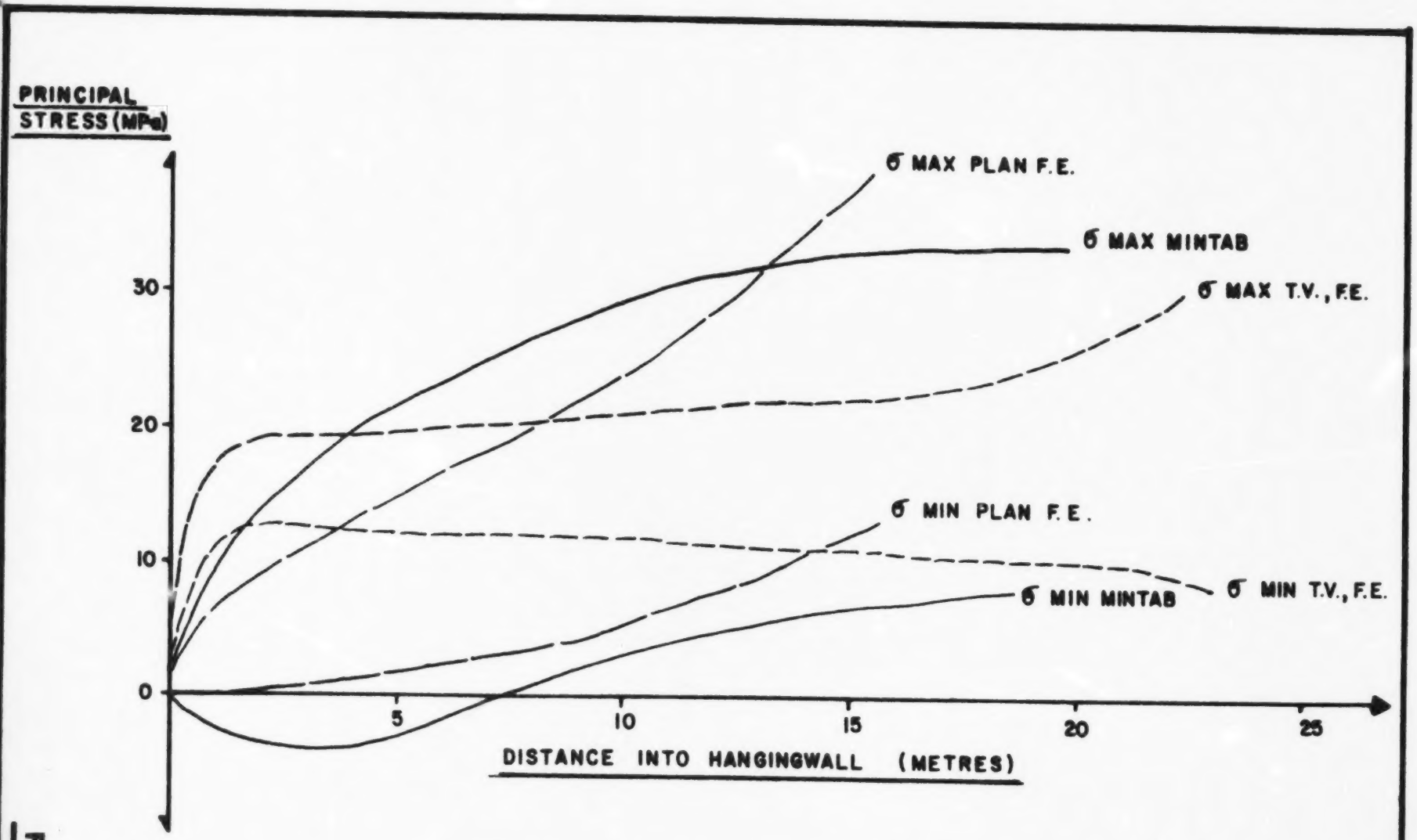
FIG: 43

No.3 ZONE TRANSVERSE F.E. MODEL

CASE 7 - REDUCED ROCK STRENGTHS

1st PANEL OPEN K = 1.5 SCALE 1 = 40

FIG: 43

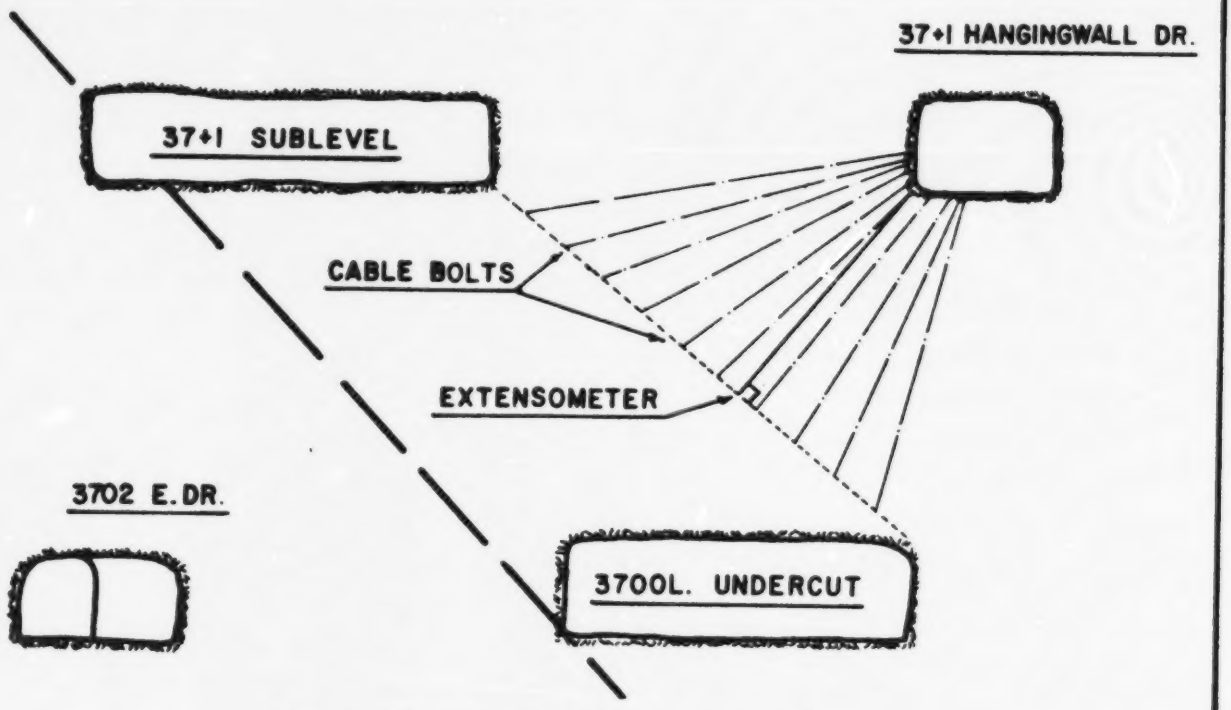



COMPARISSON OF HANGINGWALL STRESS LEVELS

WITH NUMERICAL METHOD

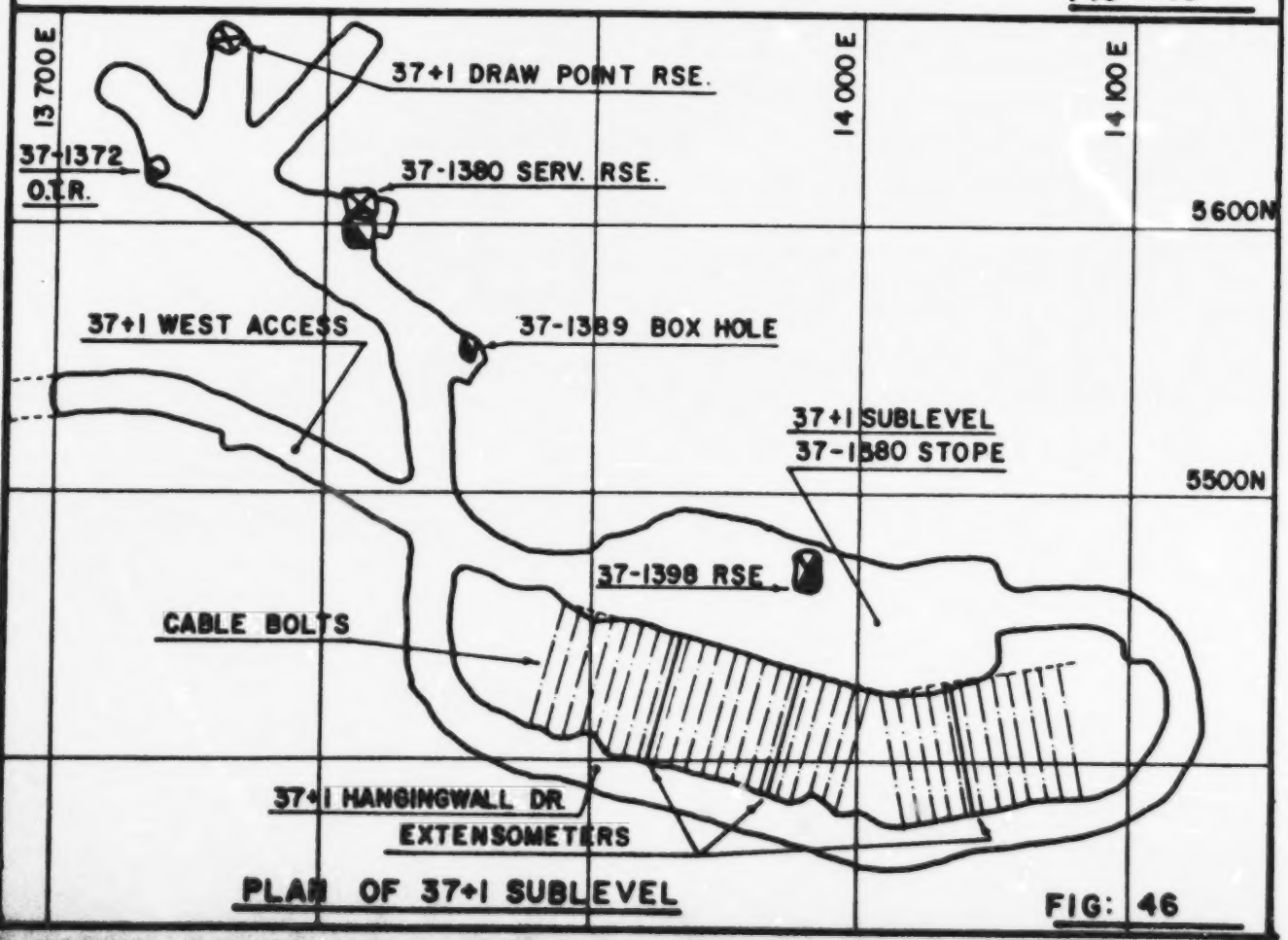
FIG: 44

FIG: 44



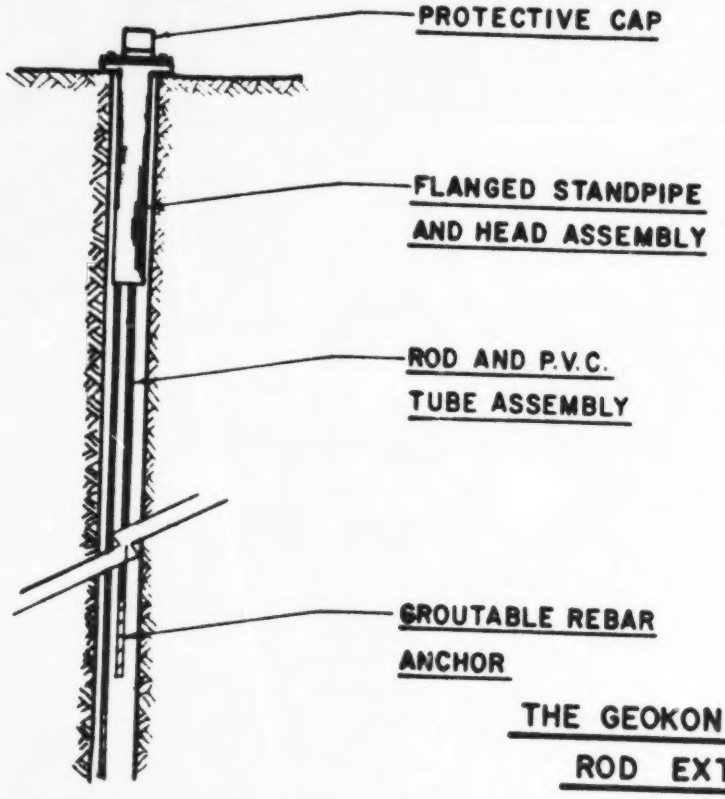

SECTION CABLEBOLT LAYOUT

FIG : 45




PLAN OF 37+1 SUBLEVEL

FIG: 46

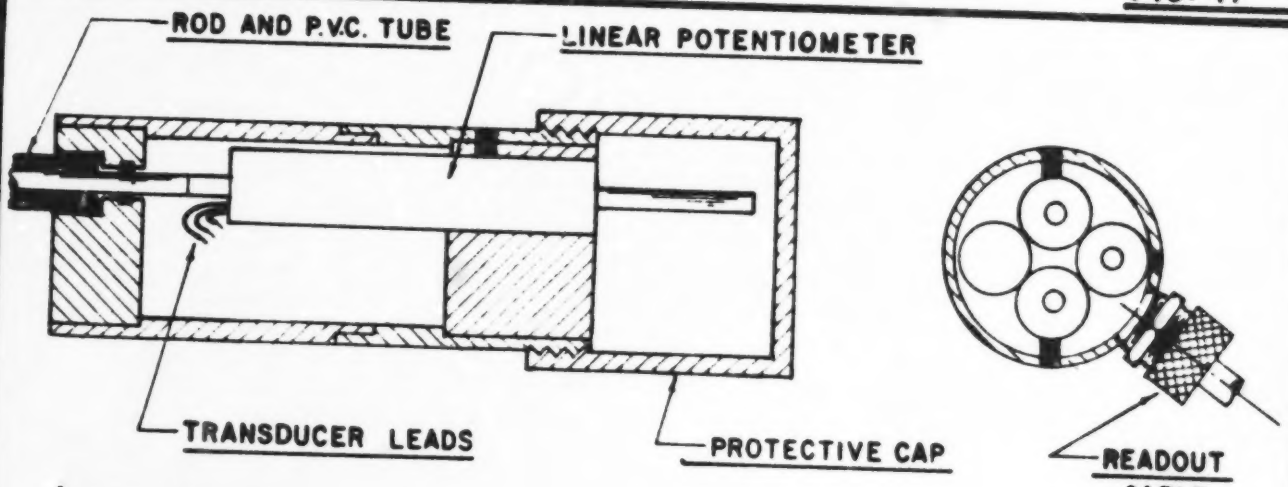



**SPECIFICATIONS**

| | |
|---|---|
| MEASUREMENT POINTS | 1 to 6 |
| RANGE | 300mm |
| MIN. READING | .01 mm |
| MIN. BOREHOLE DIAM. | 75mm |
| MAX LENGTH | 100m |

THE GEOKON MODLE "3" BOREHOLE ROD EXTENSOMETER

FIG: 47




**SPECIFICATIONS**

| | |
|---|---|
| RANGE | 100mm |
| LINEARITY | 1% |
| REPEATABILITY | 0.1% |
| ACCURACY | 0.5% |
| RESOLUTION | .01mm. |

REMOTE READOUT HEAD ASSEMBLY WITH LINEAR POTENTIOMETER

FIG: 48

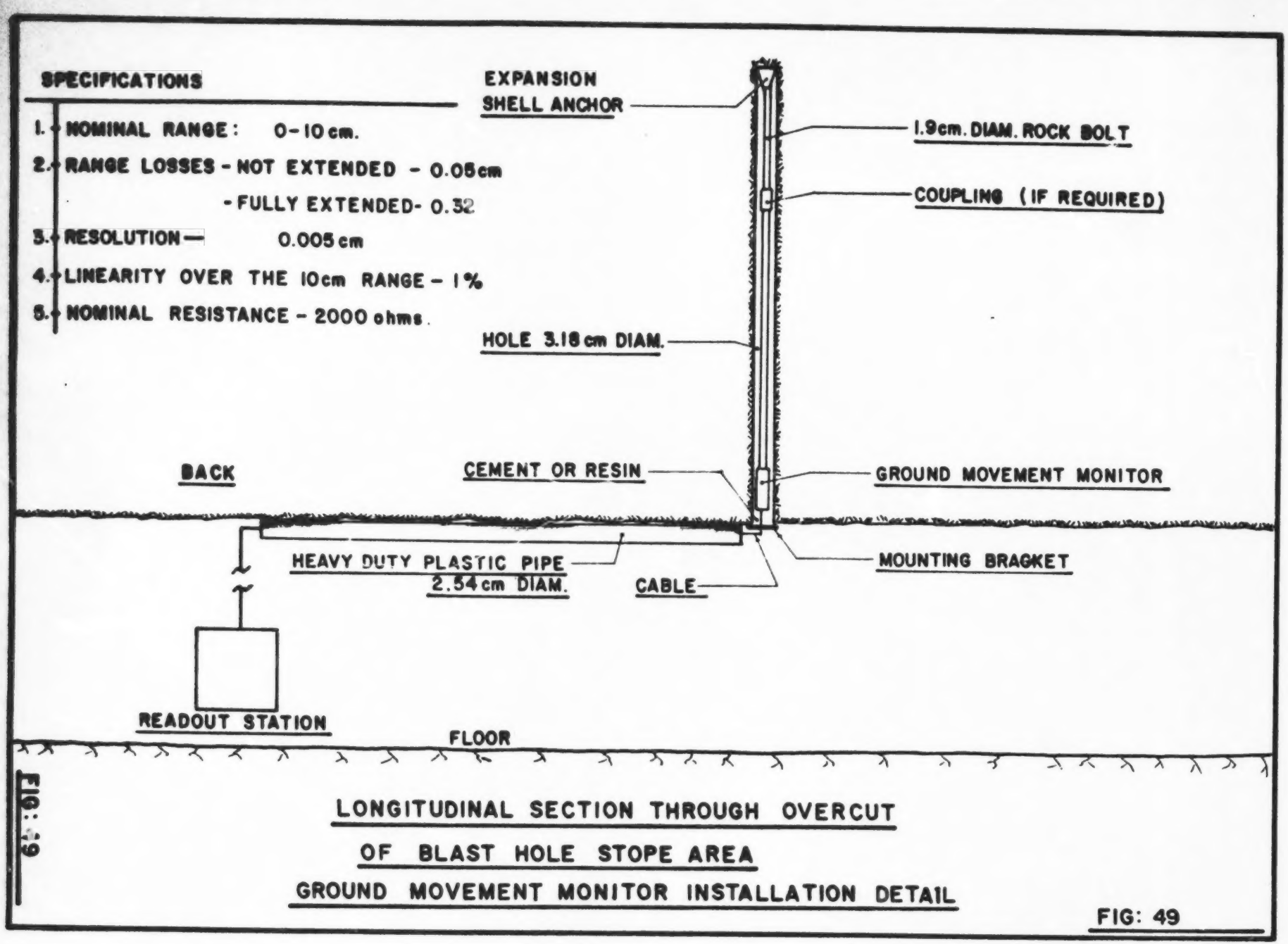


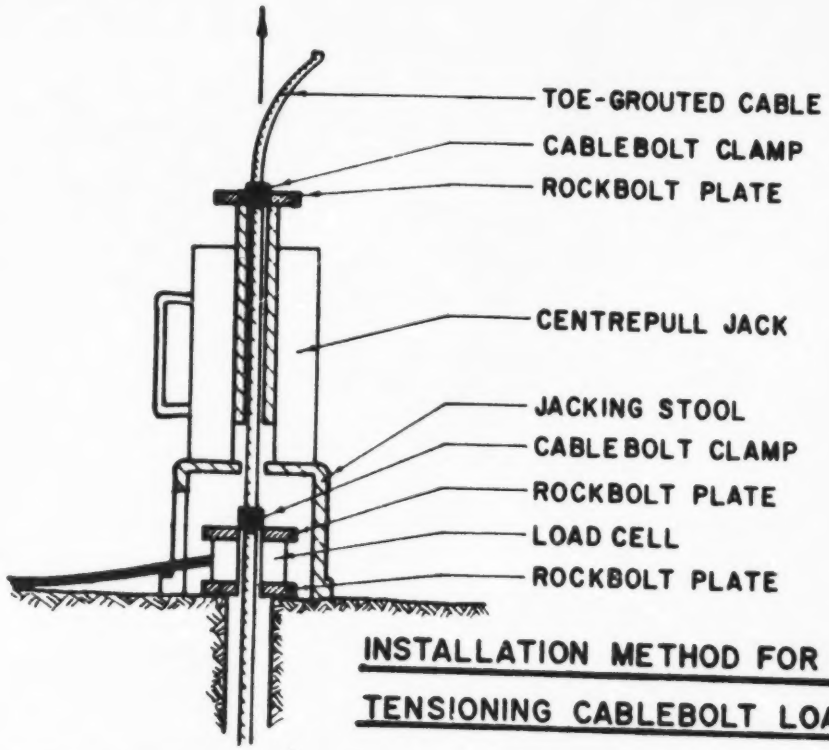

FIG: 49

LONGITUDINAL SECTION THROUGH OVERCUT

OF BLAST HOLE STOPE AREA

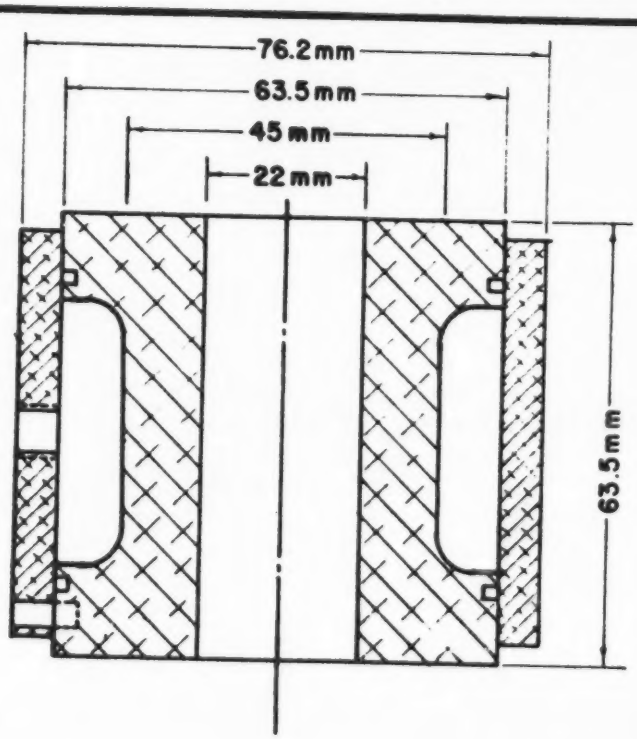
GROUND MOVEMENT MONITOR INSTALLATION DETAIL

FIG: 49




INSTALLATION METHOD FOR NOMINALLY TENSIONING CABLEBOLT LOADCELLS

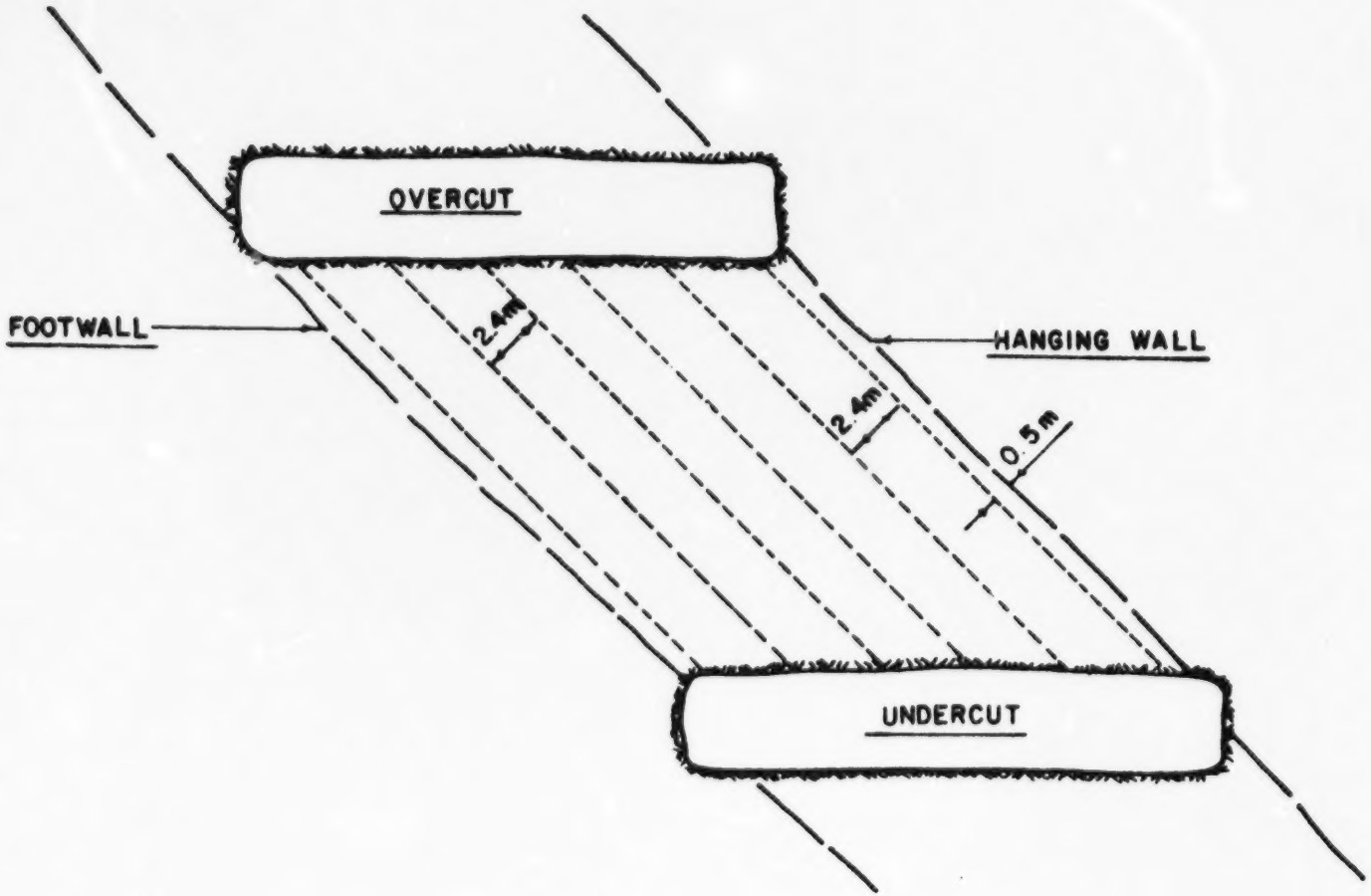
FIG: 50




| SPECIFICATIONS | |
|---|---|
| BODY AND OUTER CASE | — ALUMINUM |
| CROSS-SECTIONAL AREA | — 12.9 $cm^2$ |
| SENSORS | — 4 STRAIN GAUGES AT 90° IN SERIES - PARALLEL |
| MAX. LOAD | — 27 tonnes |
| MAX. STRESS | — 207 MPa. |
| MAX. STRAIN | — 3000$\mu\Sigma$ |
| MIN. SENSED LOAD | — 90 Kg |
| MIN. SENSED STRAIN | — 40 $\mu\Sigma$ |

STRAIN GAUGED CABLEBOLT LOADCELL

FIG: 51

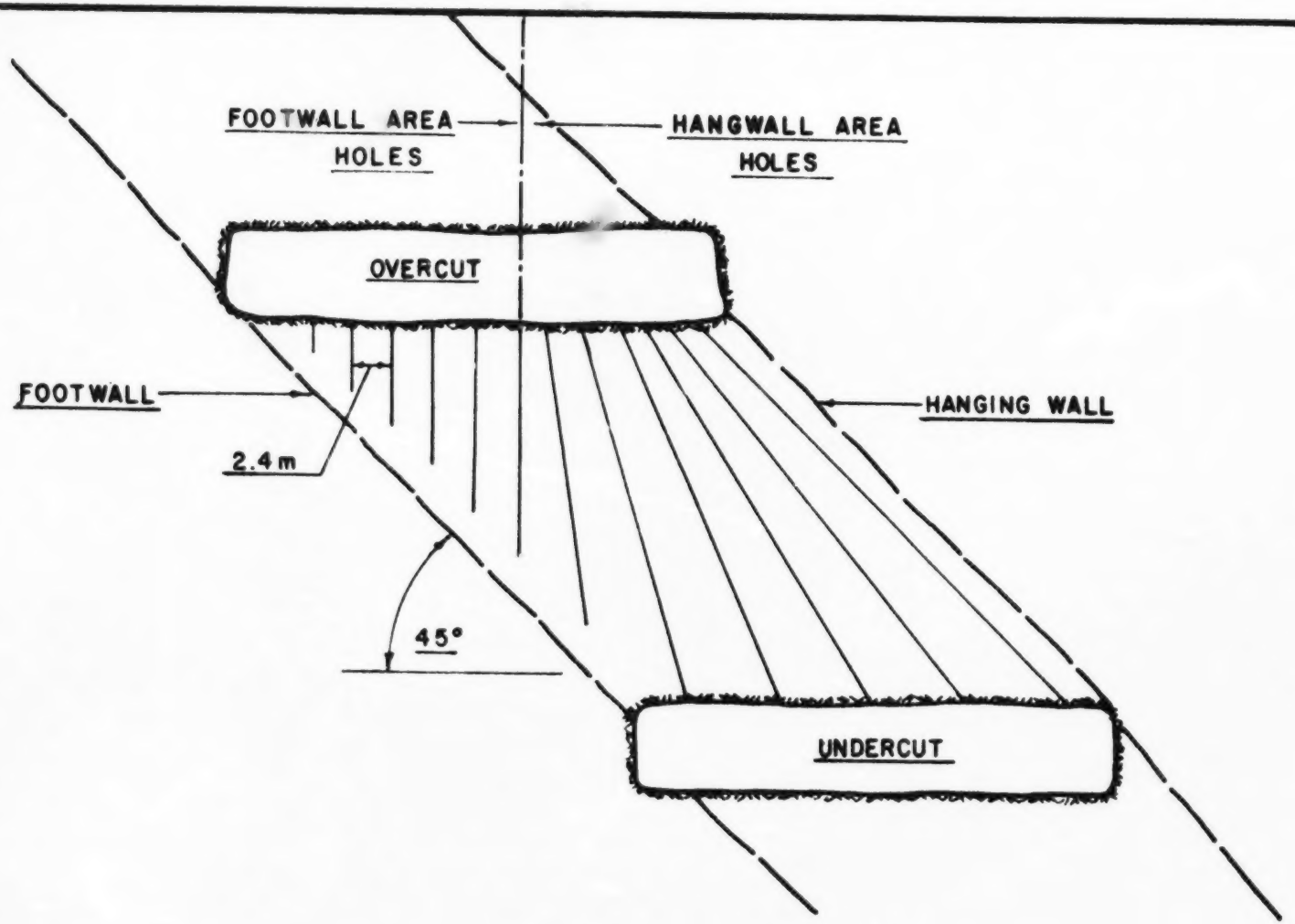



TYPICAL TRANSVERSE SECTION

SHOWING DRILLING LAYOUT USING PARALLEL HOLES

FIG: 52

FIG: 52

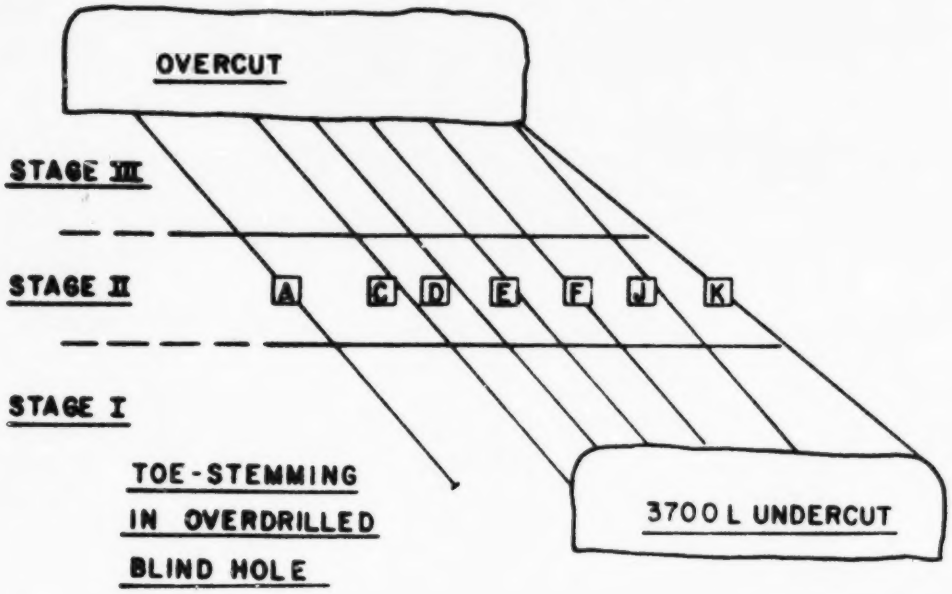



TYPICAL TRANSVERSE SECTION

SHOWING DRILLING LAYOUT USING FANNED HOLES

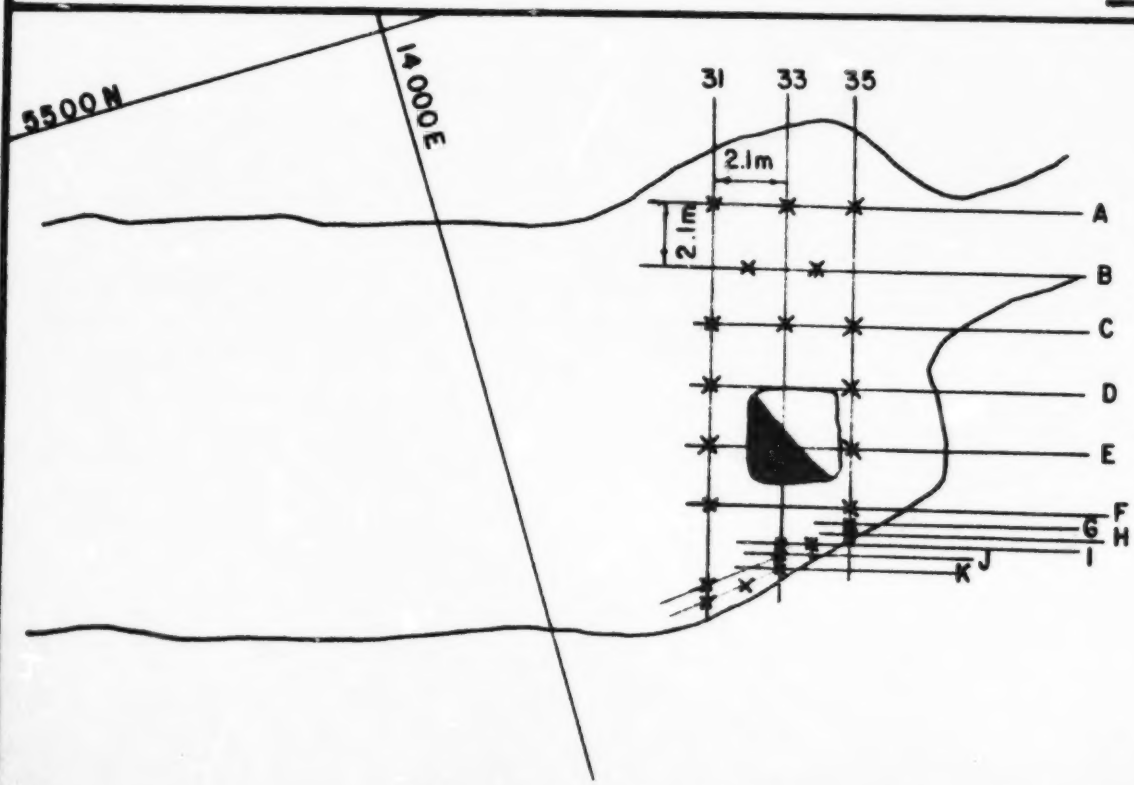
FIG: 53

FIG: 53



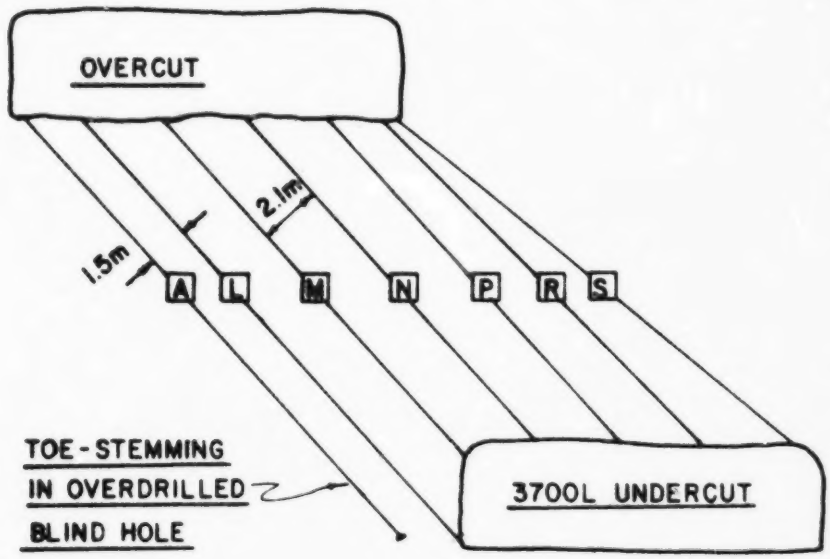

SLOT BLAST SECTION LINE 31

FIG: 54



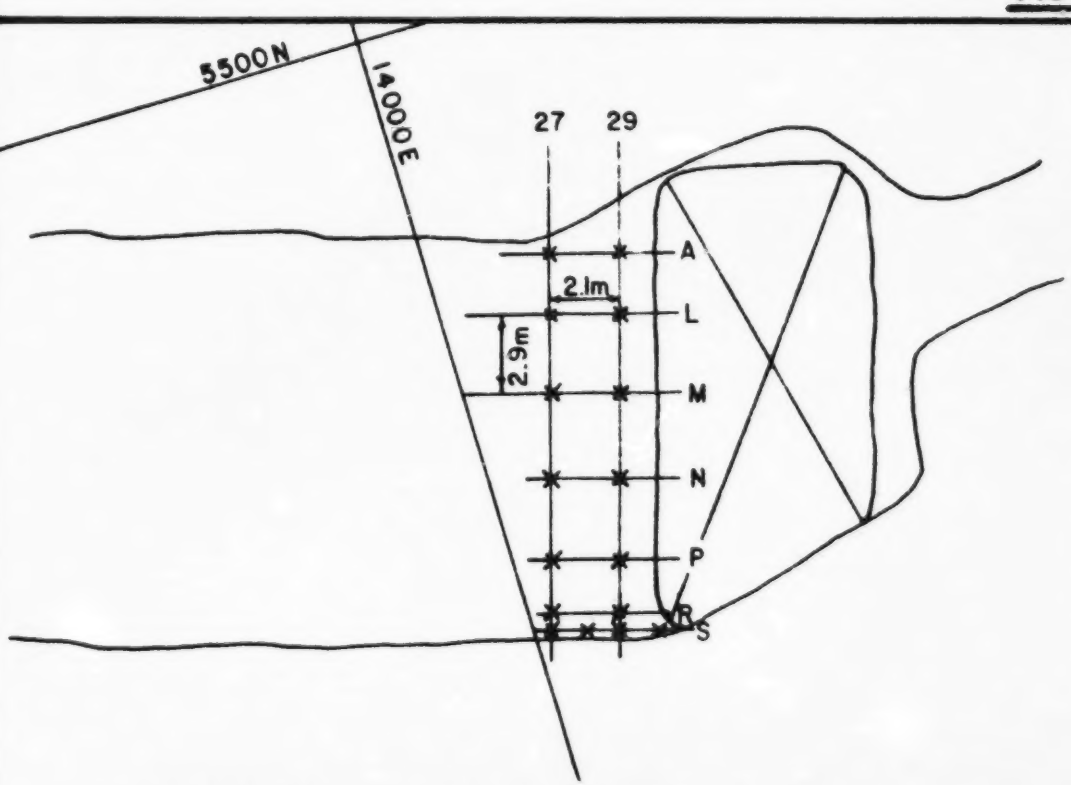

SLOT BLAST- PLAN OF OVERCUT HOLE LAYOUT

FIG: 55



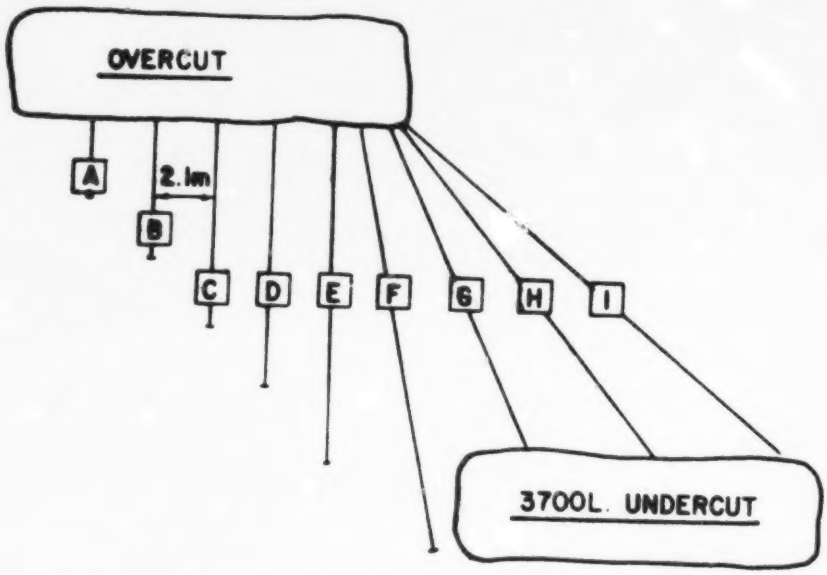

PRODUCTION BLAST - PARALLEL HOLES - SECTION LINE 29

FIG: 56



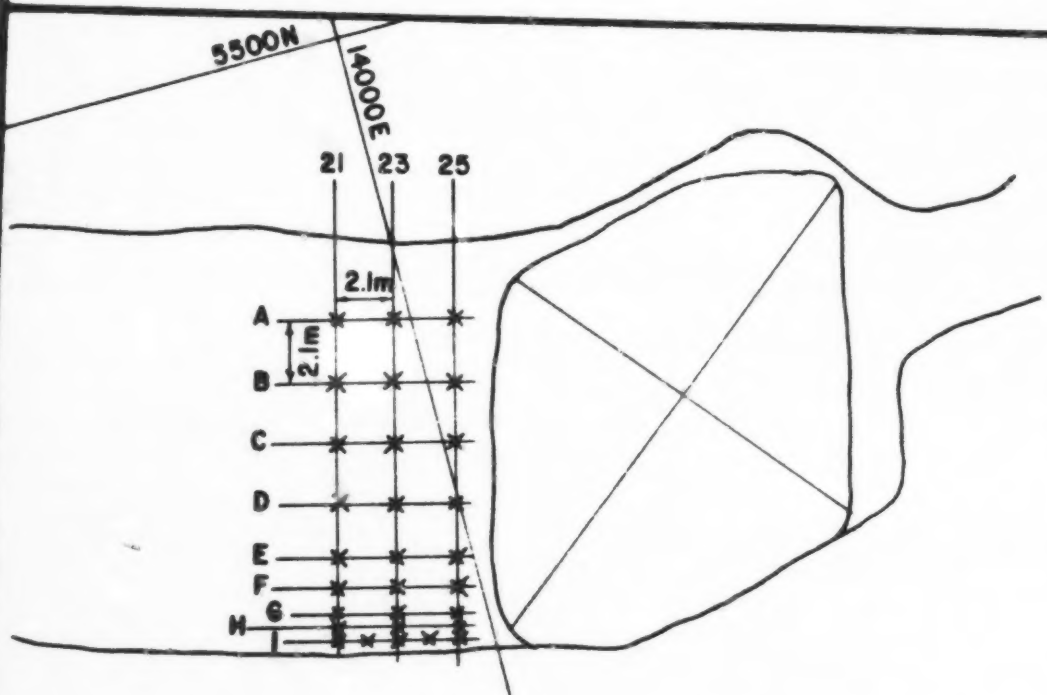

PRODUCTION BLAST - PARALLEL HOLES
PLAN OF OVERCUT HOLE LAYOUT

FIG: 57




PRODUCTION BLAST - FANNED HOLES - SECTION LINE 25

FIG: 58




PRODUCTION BLAST - FANNED HOLES

PLAN OF OVERCUT HOLE LAYOUT

FIG: 59